REGIONAL POTASH MINE STABILITY REPORT

Potash Corporation of Saskatchewan Mining Limited

ASSESSMENT OF POSSIBLE PROBLEMS IN REGIONAL MINE STABILITY WITH FUTURE MINING OF SASKATCHEWAN POTASH

F I N A L R E P O R T

BY

POTASH CORPORATION OF SASKATCHEWAN MINING LIMITED

Work on this project was conducted under the auspices of the Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources, Canada.

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EXECUTIVE SUMMARY

OF THE REPORT ON

"ASSESSMENT OF POSSIBLE PROBLEMS IN REGIONAL MINE STABILITY WITH FUTURE MINING OF SASKATCHEWAN POTASH"

The Potash Corporation of Saskatchewan Mining Limited (PCS Mining) jointly with Central Canada Potash (CCP) and Noranda Research Centre (CRN) submitted a proposal dated 19 September, 1983 in response to the Department of Supply and Services Canada (DSS) Request for Proposal RFP 26SQ. 23440-3-9125 dated 26 August, 1983, titled "Assessment of Possible Problems in Regional Mine Stability with Future Mining of Saskatchewan Potash". The joint proposal of PCS Mining, CCP and CRN was accepted by DSS and contract No. OSQ83-00246 was issued to PCS Mining by DSS on 16 January, 1984. This Final Report is submitted in compliance with the terms of the contract.

Since the inception of potash mining in Saskatchewan in 1958, large areas underground have been opened up and subsidence has occurred over the mines. In connection with subsidence, seismic events have occurred at or in close proximity to several of the mines. The earth tremors have been attributed to brittle failure of rock formations above the mining horizon. Sudden failure of these rock formations and associated seismicity could have significant consequences for the stability of the immediate roof and the safety of mine, including the possibility that it might spread across individual mine boundaries, thereby endangering nearby operations.

The methodology used to provide the vast wealth of information and experiences from the industry was to draft a series of questionnaires to determine the quality and type of data available at each participating mine pertinent to each of the problem areas under investigation. The questionnaire focused on the following topics:

a) Subsidence

b) Earthquake and water inflows

c) Historical mine development.

The data extracted from the questionnaires and also from direct interviews, were compiled and analyzed to formulate the definite statements of the problems encountered by the potash industry and to identify areas of required research and development to resolve or to alleviate the specific concerns.

The main areas of discussion are summarized as follows:

1. Shaft stability is considered to be the number one concern of the mines. The central component of shaft stability is the shaft lining and its long term structure, strength and integrity. The report discusses various possible causes of shaft instability and identifies the main areas of research required to improve stability.

2. Mining induced seismicity considers the probable causes of this phenomenon - mining pattern, extraction ratio, the rate of mining and subsidence.

3. Geological anomalies. The anomalous structures contributing to the instability - washouts, leached zones, collapse structure and Winnipegosis mounds, are discussed.

4. Water inflow. The report examines the potential risk from formation waters and the threat from the overlying water bearing formations: Dawson Bay, Souris River, Blairmore, etc. The basic design and layout of the potash mine as a preventive factor in water inflow to the mine is also addressed.

5. Brine injection wells. Under this topic, the installation of the wells is discussed in relation to the mining and environmental concern regarding these wells. The report examines the same fundamental questions about these wells and their possible threat to the instability of the mine workings. In addition, the question of the monitoring of the injection wells is outlined in the report, and the necessary steps required to appraise the data base for pressures of injection and chemical composition requirements are discussed.

6. Potash exploration wells. The drilling and abandonment practices in relation to these wells, the reliability of the cement plugs, the isolation of a safety zone surrounding the projected exploration boreholes are the subjects of this discussion. The report also examines the possible hazard poised by abandoned wells in relation to the factors listed.

Recommendation

Research and development in the above six areas has been addressed in detail and references from the work conducted in the various areas have been enlisted.

The general consensus of the various participants in the project is that the project has been successful in achieving its objective within the limits imposed by the quantity and quality of the geotechnical data available. The project has identified specific areas where research and development are required to ensure long term regional mine stability in the Saskatchewan potash mining industry. Areas where insufficient data are available, to determine if a problem does exist or may develop in the future, have been identified and thus requirements for further research and probable development of instruments and/or techniques have been outlined.

The second part of the report deals with the problems of subsidence due to mining of potash and describes the use of an empirical subsidence prediction method applied to Saskatchewan potash mines. Subsidence data were collected from five producing potash mines in southern Saskatchewan.

A review of existing subsidence prediction methods was made and applicability of the methods was considered. The data indicate that subsidence tends to be time dependent. The plastic nature and low strength of the evaporites cause subsidence to occur in two stages; an initial contribution occurs at some time after first mining, followed by a constant subsidence rate. The functions relating initial and steady state subsidence of a point overlying an excavation were found to correlate to extraction through an influence function. A computer program was developed to predict subsidence using a zone area method of influence function. The results show reasonable correlation to field measurements.

RÉSUMÉ du rapport: Évaluation des problèmes potentiels de stabilité régionale associés aux futures exploitations minières de potasse en Saskatchewan.

Potash Corporation of Saskatchewan Mining Limited (PCS Mining) conjointement avec Central Canada Potash (CCP) et le Centre de Recherche Noranda (CRN) a soumis une proposition le 19 septembre 1983, concernant la demande de soumission no. FRP 26SQ 23440-3-9125 du Ministère des Approvisionnements et Services Canada (MASC) datée du 26 août 1983 et intitulée: Évaluation des problèmes potentiels de stabilité régionale associés aux futures exploitations minières de potasse en Saskatchewan. La proposition conjointe de PCS Mining, CCP et CRN a été acceptée par le Ministère, et le 16 janvier 1984, PCS Mining recevait le contract no. OSQ83-00246. Ce rapport final est soumis en accord avec les termes du contrat.

Depuis le début des mines de potasse en Saskatchewan en 1958, de vastes ouvertures souterraines ont été excavées et des affaissements de terrain au dessus des exploitations minières sont survenus. Des secousses telluriques en relation avec ces affaissements ont été enregistrées à plusieurs sites miniers, ou à proximité de ceux-ci. Les tremblements de terre ont été attribués aux bris de formations rocheuses au-dessus des chantiers miniers. Ces bris soudains de formations rocheuses et ces activités telluriques peuvent avoir de sérieuses conséquences pour la stabilité des toits et la sécurité de la mine en général, sans oublier la possibilité que ces conditions peuvent s'étendre au delà des limites de propriété individuelle et mettre en danger les exploitations avoisinantes.

Une série de questionnaires adaptés à chacun des problèmes sous investigation ont été envoyés à chacune des mines participantes afin de pouvoir assembler la vaste richesse d'information et d'expérience répartie à travers l'industrie. Les questionnaires étaient orientés sur les aspects suivant:

a) Affaissement des toits
b) Tremblement de terre et infiltration d'eau
c) Développement historique

L'information recueillie des questionnaires et des entrevues était compilée et analysée afin de pouvoir définir précisement les problèmes affrontés par l'industrie de la potasse et d'identifier les secteurs requérant plus de recherche et de développement afin de solutionner ou de réduire certaines inquiétudes particulières.

Les principaux sujets de discussion sont résumés comme suit:

1. La stabilité du puits est considérée comme étant le problème numéro un, commun à toutes les mines. La solidité et l'intégrité structurale du recouvrement du puits constituent les facteurs les plus critiques. Le rapport mentionne plusieurs causes possibles de l'instabilité du puits et identifie le secteur recherche principal.
2. Les patrons d'extraction, les taux de minage ainsi que les affaissements sont considérés comme les causes probables des activités telluriques.
3. Anomalies géologiques. Les structures anomales contribuant à l'instabilité, telles que: zones délavées ou dissoutes, effondrements et monticules Winnipegois sont également discutées.
4. Infiltration d'eau. Le rapport examine les risques potentiels associés aux formations d'eau et les menaces provenant des couches phréatiques telles que les formations Dawson Bay et Blairmore, etc. Les concepts de bases ainsi que la disposition générale des mines de potasse sont présentés comme étant des facteurs préventifs aux infiltrations d'eau.
5. Puisards d'injection de saumure. Sous ce thème, l'installation des puisards est discutée en relation avec leurs effets sur l'environnement et la mine. Le rapport examine la même question fondamentale concernant ces puisards et leurs effets possibles sur la stabilité des chantiers miniers. De plus, le rapport souligne la question de surveillance des puisards ainsi que les procédures nécessaires à l'interprétation des informations concernant la pression d'injection et la composition chimique des saumures.
6. Trous d'exploration. Le forage et les procédures d'abandon de ces trous d'exploration ainsi que la fiabilité des bouchons et l'isolement à l'aide d'un pilier entourant les trous d'exploration sont les thèmes de cette discussion. Le rapport étudie également les risques possibles causés par les trous abandonnés en relation avec les facteurs déjà mentionnés.

Recommandations

Les secteurs requérant le plus de recherche et de développement parmi les 6 points ci-haut mentionnés, ont été élaborés en détail, en référence avec le travail qui a déjà été accompli.

Les participants à ce project sont d'avis qu'il a été un succès en rencontrant les objectifs fixés.

Le project a identifié les secteurs où le plus de recherche et de développement sont nécessaires pour assurer la stabilité minière régionale à long terme de l'industrie de la potasse en Saskatchewan. Les secteurs ou l'information disponible est insuffisante pour déterminer si un problème existe ou peut se développer dans le futur, ont été identifiés. Les points nécessitant le plus de recherche et probablement de développement de nouveaux instruments et/ou de nouvelles techniques, ont été soulignés.

La seconde partie du rapport traite des problèmes d'affaissements causés par le minage de la potasse, et décrit l'utilisation des méthodes empiriques de prédiction d'affaissements employées par les mines de potasse de Saskatchewan. L'information concernant les affaissements proviennent de 5 mines de potasse opérant dans la partie sud de la province.

Une revue des méthodes de prédiction d'affaissements a été faite et l'application possible de ces méthodes a été considérée. L'information disponible indique que les affaissements tendent à être en fonction du temps. La nature plastique et la faible résistance des évaporites font que les affaissements se manifestent en deux étapes; l'étape intiale se manifeste quelques temps après la première extraction, par la suite l'affaissement se produit à un taux constant. Les essais de corrélation de l'affaissement d'un point localisé à la surface avec le pourcentage d'extraction utilisant une fonction d'influence se sont avérés un succès. Un programme pour ordinateur utilisant la fonction d'influence a été développé pour prédire l'affaissement. Les résultats obtenus démontrent une corrélation raisonnable avec les mesures effectuées sur place.

VOLUME ONE

ASSESSMENT OF POSSIBLE PROBLEMS IN REGIONAL MINE STABILITY WITH FUTURE MINING OF SASKATCHEWAN POTASH

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1. INTRODUCTION

The Potash Corporation of Saskatchewan Mining Limited (PCS Mining) jointly with Central Canada Potash (CCP) and Noranda Research Centre (CRN) submitted a proposal dated 19 September, 1983, in response to the Department of Supply and Services Canada (DSS) Request for Proposal RFP 26SQ. 23440-3-9125 dated 26 August, 1983, titled "Assessment of Possible Problems in Regional Mine Stability with Future Mining of Saskatchewan Potash". The joint proposal of PCS Mining, CCP and CRN was accepted by DSS and contract Serial No. OSQ83-00246 was issued to PCS Mining by DSS on 16 January, 1984. This Final Report is submitted in compliance with the terms of the contract.

1.1 BACKGROUND ON THE PROJECT

Since the inception of potash mining in Saskatchewan in 1958, large areas underground have been opened up and subsidence has occurred over the mines. In connection with subsidence, seismic events have occurred at or in close proximity to several of the mines. The earth tremors have been attributed to brittle failure of rock formations above the mining horizon. Sudden failure of these rock formations and associated seismicity could have significant consequences for the stability of the immediate roof and the safety of mine, including the possibility that it might spread across individual mine boundaries, thereby endangering nearby operations.

In addition, the effectiveness of the salt cover as a water barrier could be destroyed due to subsidence and the development of fractures which could provide new pathways for water to enter mine workings. (From: Appendix "A", Statement of Work, Contract Serial No. OSQ83-00246).

1.2 OBJECTIVE OF THE PROJECT

The Canada Centre for Mineral and Energy Technology (CANMET) defined the objective to be the establishment of the geotechnical data base required to assess possible long term regional stability problems in Saskatchewan potash mining and the potential benefits (higher extraction ratio, improved productivity) to be achieved using backfill or other options. (From: Appendix "A" of the contract).

1.3 WORK SCOPE AS DEFINED BY CANMET

The workscope of the project was defined in Appendix "A" of the Contract to be:

1. Identify long term problems of regional mine stability based on projected production plans and adverse effects observed until now.

2. Document case histories from engineering data at minesites which show the relationship between surface subsidence and parameters such as mining advance, extraction ratio, depth of mining, geological type of overburden, height and size of mined-out areas and elapsed time.

3. Estimate the anticipated surface subsidence based on the projected production plans, by using a suitable analytical method (e.g. "Incremental Design in Rock Mechanics" by D.F. Coates and M. Gyenge).

4. Quantify the adverse effects on the safety of operations in relationship to the anticipated surface subsidence.

5. Identify and evaluate options for economically viable mining layouts to reduce long term adverse effects on regional mine stability.

6. Identify the effects of salt backfill in terms of local and regional mine stability, surface subsidence and improved exploitation of potash resources.

7. Identify further research required to address the concern of required mine stability with time.

1.4 PARTICIPANTS IN THE PROJECT

The project was initiated and funded by CANMET. The participants in the project, on a joint venture basis, were:

a) Earth Sciences Section, Engineering Department of PCS Mining, as Project Leader and Coordinator

b) the four potash mines operated by PCS Mining, specifically:

PCS Mining, Allan Division (Allan)

PCS Mining, Cory Division (Cory)

PCS Mining, Lanigan Division (Lanigan)

PCS Mining, Rocanville Division (Rocanville)

c) Central Canada Potash Division of Noranda Mines Limited (CCP)

d) Noranda Research Centre, Point Claire, Quebec (CRN)

1.5 METHODOLOGY USED IN THE PROJECT

A committee was set up to oversee the project to its conclusions. The committee consisted of the following:

Senior Scientist, Earth Sciences, PCS Mining as Chairman and Project Leader

Chief Mine Engineer (and one alternate) from each of the five participating mines as Members

Project Coordinator/Analyst (from Earth Sciences, PCS Mining)

The committee held an initial meeting to set out the overall terms, guide lines, methodology and objectives of the project. This meeting finalized the list of problems the mines considered to be affecting mine stability in the potash industry. It then held a second meeting half-way through the project to review progress to date and to finalize the completion of the project. A final meeting was held to review and approve the completion of the project and the final report.

The methodology agreed upon was to draft a series of questionnaires to determine the type and quantity of data available at each of the mines pertinent to each of the problem areas to be investigated. These questionnaires were then sent to the mines for completion by them. The Project Coordinator/Analyst then interviewed the respondents for clarification of their replies where it was necessary for accuracy or completeness of the data. The questionnaires were focused on the following topics:

a) Subsidence

b) Earthquakes and Water Inflows

c) Historical Mine Development

A blank copy of each of the questionnaires is included as Appendix A to this report.

The data contained in the completed questionnaires and from the direct interviews were then compiled and analyzed to formulate the definitive statements of the problems encountered by the potash industry concerning mine stability, and to identify those areas where research and development is required to resolve or alleviate the concerns of the mines.

In conjunction with the questionnaires and direct interviews, a literature search was carried out, focused on the stability problems identified by the mines. The literature generated from this search was then reviewed for its applicability to the project.

The detailed investigations of subsidence experienced to date by the mines, as documented by available data, and the suitability of analytical and numerical modelling methods to estimate anticipated surface subsidence (Items 2 and 3 of CANMET's workscope) were assigned to Noranda Research Centre, utilizing the data generated by the Subsidence questionnaires.

The results of the survey to determine the problems affecting mine stability, as indicated by the various mines, and the recommendations for research and development (Items 1, 4, 5, 6 and 7 of the workscope) are reported in Volume I of this report. Volume II of this report refers entirely to the results of the studies on subsidence carried out by Noranda Research Centre.

1.6 LIMITATIONS OF THE PROJECT

During the course of this project, it became apparent that there were certain operational and physical limits that would affect the ultimate completeness and thus, the absolute achievement of the objectives of this project.

The operational limits were the actual or perceived requirements for CONFIDENTIALITY of some of the data requested. It should be emphasized here that the potash fertilizer industry is very competitive, both in Canada and world wide, and it follows that the various producers are extremely sensitive to requests for certain specific data concerning their operation. Such requests for information concerning potash grades, carnallite content, insolubles content, future production plans, detailed rock mechanics data and instrumentation were not always provided or were requested to be kept confidential. As a result, the project methodology concentrated on determining <u>if</u> this information was available at the individual mines, rather than attempting to collect the specific data.

The physical limitations to the completeness of the information was the factual lack of the information or data requested. This is, in part, due to the fact that the individual mines have developed with independent management over the years, with differing geological conditions and consequent different rock mechanics concerns and programs. The methodology concentrated on determining the type and quantity of information available for future research use.

2. GEOLOGICAL SETTING

2.1 HISTORICAL DEVELOPMENT

Potash was first recognized in Saskatchewan in core from an oil well drilled by Imperial Oil Limited in August, 1942, approximately 110 km south of Regina. The potash occurred at a depth of 2 330 m, too deep for conventional underground mining.

The first commercial production of potash in Saskatchewan was attempted in 1951 near Unity. Ten years later, the shaft, which had reached a depth of 558 m, was flooded. The site was sealed and abandoned in 1968 (Fuzesy 1982).

Potash Company of America was the first company to successfully complete shaft sinking and went into production in 1958. However, due to water seepage in the shaft, production was suspended and did not resume until 1965. During the period 1962-1970, eight other conventional underground mines and one solution mine were brought on-stream.

In 1975, the Saskatchewan Government set up a crown corporation, Potash Corporation of Saskatchewan (PCS), to develop a new mine at Bredenbury in the southeastern part of the Province. This project was deferred and, in its place, the Corporation purchased the mines operated by Duval, Alwinsal and Sylvite, renaming them the Cory, Lanigan and

Rocanville Divisions of Potash Corporation of Saskatchewan Mining Limited. In addition, the Potash Corporation of Saskatchewan Mining Limited purchased 60% interest in the Allan Potash Mine from US Borax and Swift Canadian Company (40% interest is owned by Texasgulf, now Kidd Creek Mines) renaming it the Allan Division. A further purchase of potash reserves, located in the International Minerals and Chemical Corporation (IMC) mine at Esterhazy, was concluded with Southwest Potash (AMAX). These reserves are mined by IMC for the Corporation under a production agreement. Figure 1 shows the location of the producing mines in the Province of Saskatchewan.

2.2 GEOLOGY OF THE POTASH

The Saskatchewan deposits occur in essentially flat-lying beds near the top of a thick sequence of halite and anhydrite, more than 200 m thick in places, known as the Prairie Evaporite. The potash occurs at a depth of approximately 1 000 m in the conventional underground mining areas of the Province. The Prairie Evaporite is in the earliest or lowest of three evaporite cycles in the Middle Devonian formations of the Williston Basin. The Williston Basin is located at the southern extremity of the larger Elk Point Basin (Figure 2) which extends south-easterly from northeastern Alberta, through Saskatchewan into North Dakota, southwestern Manitoba and northeastern Montana.

The Elk Point Basin is considered to have been a great dead sea in which, due to an arid climate and intense evaporation, first anhydrite and then halite beds accumulated rapidly on top of marine limestones.

Insoluble material became increasingly common in the water and was deposited as thin seams or disseminated amongst the evaporite salts. As the result of continuing evaporation, potassium and magnesium collected in the Williston Basin and were deposited as potash. These salts were then sealed by muds of the incoming Dawson Bay sea (Fuzesy 1982). The sequence of muds (now clays) overlain by limestone then evaporite salt is considered to be a cycle (Figures 3, 4).

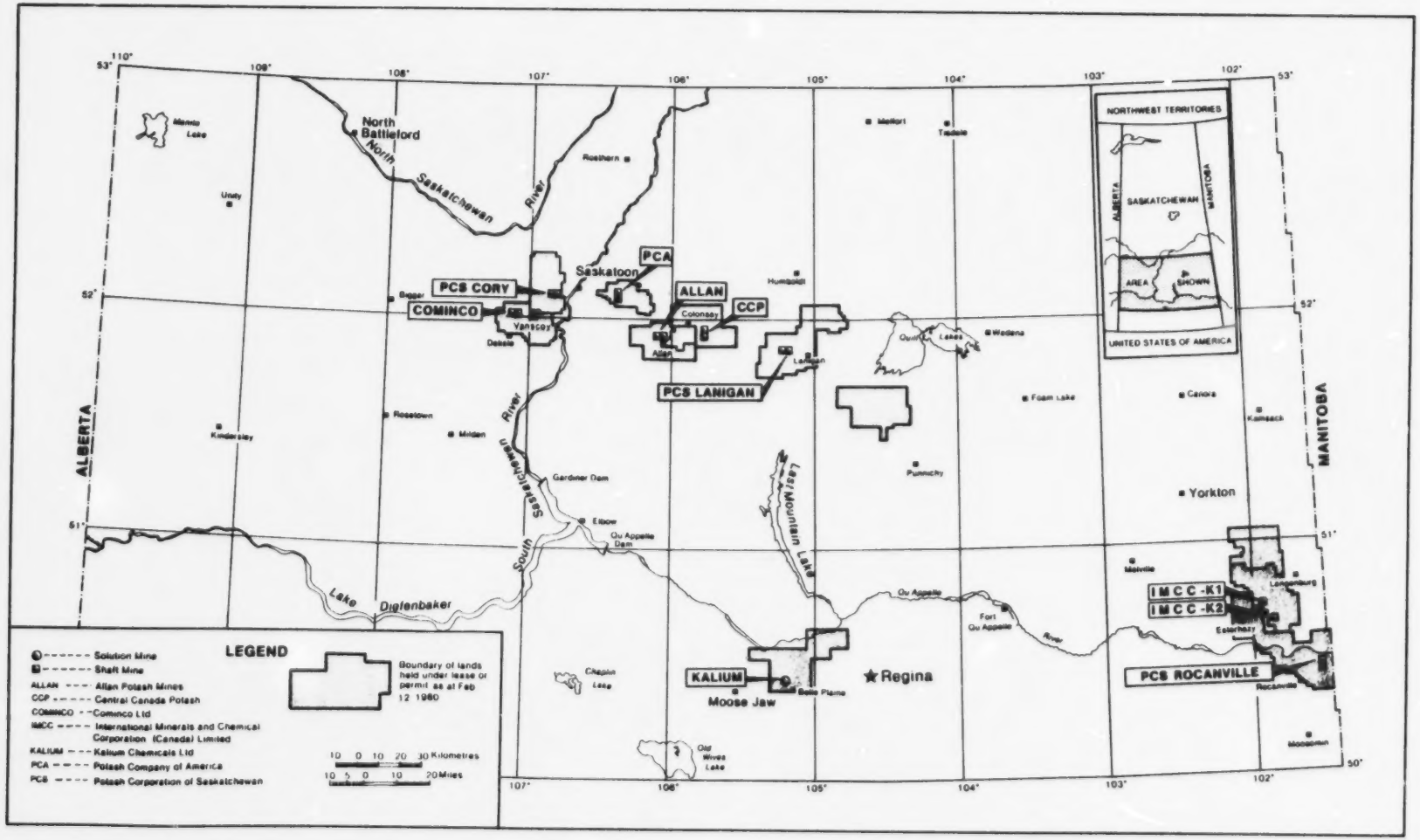


Figure 1. Potash mines, lease and permit areas in Saskatchewan (Fuzesy, 1982).

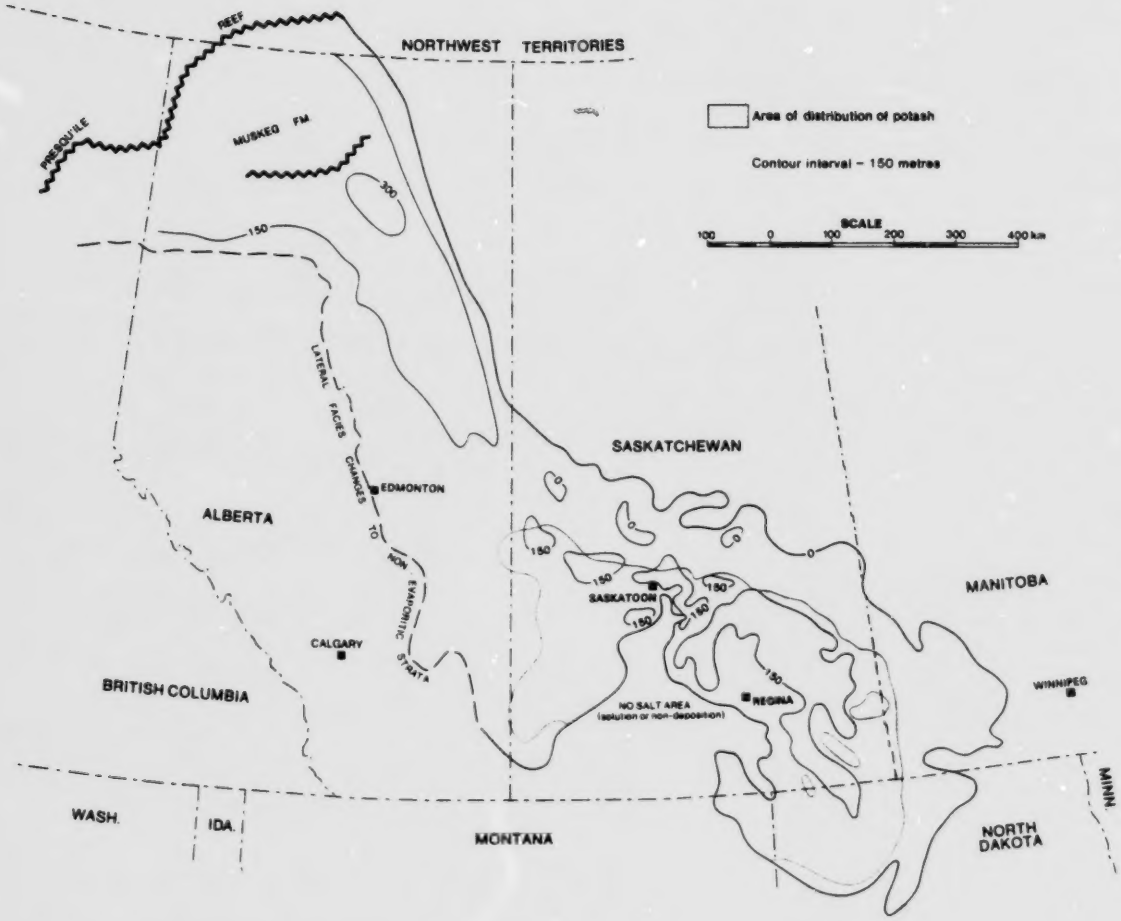


Figure 2 - Isopach map of the Prairie Evaporite in the Elk Point Basin (Fuzesy, 1982)

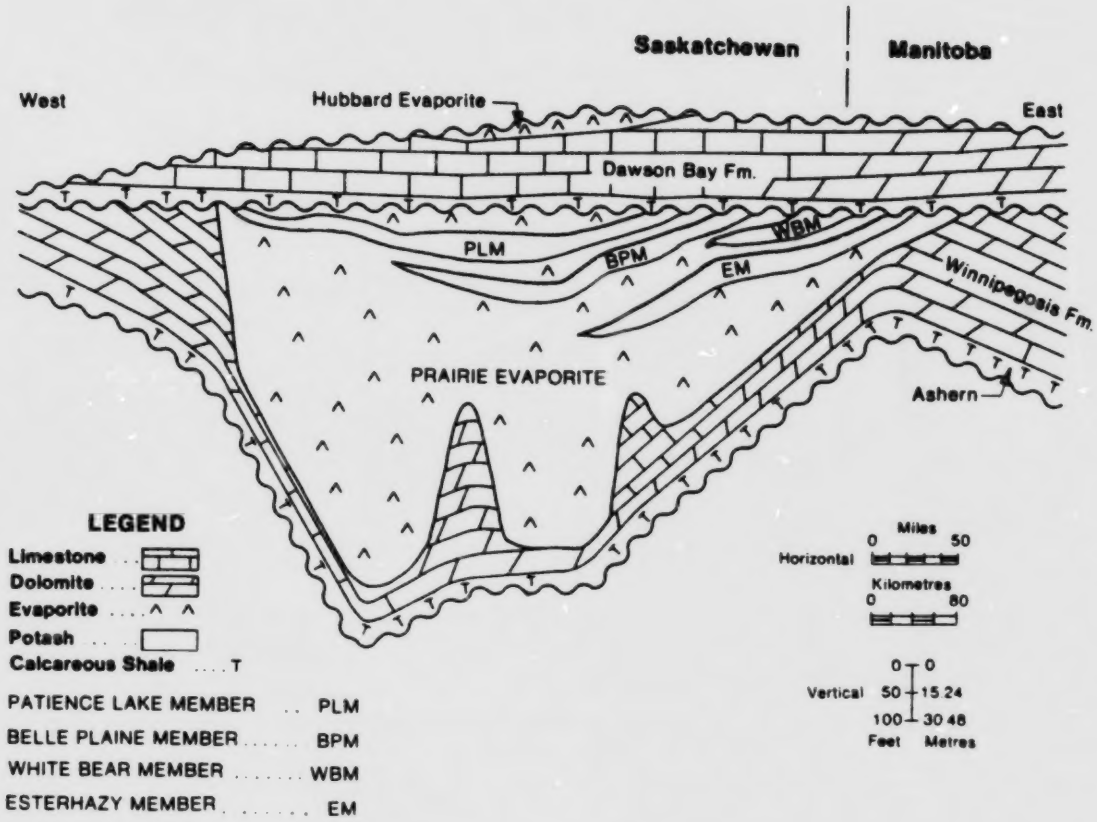


Figure 3 - Diagrammatic cross-section through the Prairie Evaporite in Saskatchewan and Manitoba (Fuzesy, 1982)

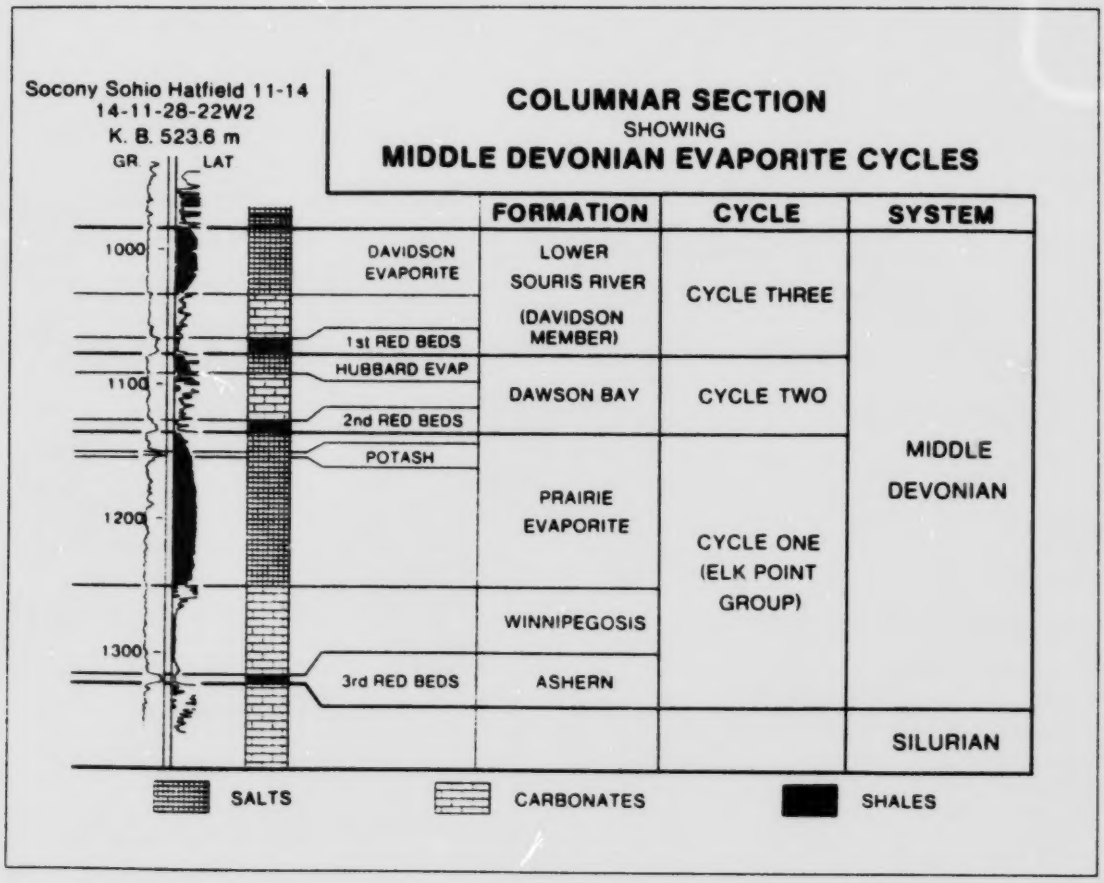


Figure 4 - Columnar section showing Middle Devonian evaporite cycles (Fuzesy, 1982)

3. PROBLEMS AFFECTING POTASH MINE STABILITY

The various meetings with the mine staff and the review of the questionnaires completed by the mines have identified the problems that the potash mines participating in the study consider to be of concern to the stability of the mines. Each problem is discussed below, detailing the past history and experience of the mines in the course of their mining operations to this time.

3.1 SHAFT STABILITY

The stability of the shafts is considered to be the number one concern of the mines. The shafts provide the only entrance to and exit from the mining level, and are thus absolutely crucial to the entire operation at each minesite. In addition to providing safe access to and from the mining level, the shafts must also be constructed to ensure that the water-bearing formations overlying the mining horizon are sealed off to prevent the formation waters, which are under pressures over 7 500 kPa (up to 1 100 psi), from entering the mine through the shaft lining and thus possibly flooding the mine. The critical component of the shaft is the shaft lining - in particular, its long term structural strength and integrity.

3.1.1 DESCRIPTION OF POTASH SHAFT LININGS

In general, potash mine shafts are lined with concrete, cast iron or steel tubbing installed at major water-bearing formations. Chemical or cement grout is used to seal off minor water-bearing formations, joints, fractures, etc., behind the concrete lining.

The concrete liner has variable thickness and strength specifications that depend on the hydrostatic pressures known to occur at the depths of the various water-bearing strata. For example, at the Rocanville mine, the concrete lining varies in thickness from 50.8 cm to 81.3 cm and compressive strength specifications of 34 475 kPa to 48 265 kPa.

The cast iron or steel tubbing varies in thickness depending on the hydrostatic pressures to be contained. At Rocanville, the tubbing varies from 6.83 cm to 9.84 cm.

3.1.2 AGE FACTOR IN SHAFT STABILITY

The chemical degradation or corrosion of the shaft lining material over time is undoubtedly the most important and most worrisome factor affecting shaft stability that categorically cannot be detected with present technology. The age of the shaft lining is therefore most important, if not <u>the</u> determining factor, when considering the overall structural integrity of the shaft.

The first successful potash mine shaft was completed in 1958 by Potash Company of America at Patience Lake, approximately 20 km east of Saskatoon. Seepage of water into the shaft led to a production halt in 1959 while a program of grouting and shaft repair was carried out and completed in 1962. In 1962, International Minerals and Chemicals Corporation completed the K1 shaft at Esterhazy. From 1967 to 1970 inclusive, 13 additional potash mine shafts were completed in Saskatchewan.

In 1979, Lanigan Division completed the sinking of its second shaft, the latest shaft completed in the province.

TABLE I

SUMMARY OF AGE OF SHAFTS

No. of Shafts	Age (in years, ± 6 months)
1	26
1	22
13	14 - 17
1	5

SOURCE: FUZESY, (1982), SASK. ENERGY AND MINES, REPORT 181

TABLE II

AGE OF POTASH SHAFTS

MINE	SHAFT NO.	COMPLETED DATE	AGE IN YEARS (± 6 MONTHS)
Allan	1	1968	16
	2	1967	17
Central Canada	1	1969	15
	2	1969	15
Cominco	1	1968	16
	2	1968	16
Cory	1	1968	16
	2	1968	16
IMC	K1	1962	22
	K2	1967	17
Lanigan	1	1968	16
	2	1979	5
PCA	1	1958	26
	2	1969	15
Rocanville	1	1970	14
	2	1970	14

SOURCE: FUZESY, 1982, SASK. ENERGY & MINES, REPORT 181

NOTE: Saskatchewan Mining Regulations require that each underground potash mine maintain at all times two independent shafts (Sask. Mines Regulations, 1978, 410.00). The accepted practice is to utilize one shaft for production, and one for service. The exception to this practice is at IMC at Esterhazy where both shafts (K1 and K2) are used for production and service.

3.1.3 POSSIBLE CAUSES OF SHAFT INSTABILITY

The major concern of the mines with respect to the shafts is that the seal, the shaft lining, is not broken, thus allowing the formation waters into the shaft and hence, into the mine. There are basically five scenarios which could lead to the structural instability of the shaft lining.

3.1.3.1 SHAFT OPERATIONAL ACCIDENT DAMAGING THE SHAFT LINING

The concern is that the strength of the shaft lining is sufficient to withstand the impact or dynamic loading of material accidentally striking the shaft lining. Such an accident could occur by:

a) human error during the lowering in the shaft of inadequately secured pieces of equipment or material, or

b) mechanical or other failure of hoist ropes or equipment resulting in the dropping of a cage or skip in the shaft.

In fact, several such accidents have occurred, but with no structural damage to the shaft linings. In 1963 and 1968, steel plate was accidentally dropped down the shaft at Lanigan. In December, 1982, the cage hoist ropes at Allan were weakened by a fire in the hoist rope lubricating equipment, with the cage breaking free of the hoist ropes and falling to the bottom of the shaft. Other mines have suffered similar accidents with no damage inflicted to the shaft lining.

The designed strength of the shaft linings is considered sufficient to withstand most anticipated accidents. However, if the material should strike a portion of the shaft lining that had been weakened by chemical degradation, serious consequences could follow from such an accident.

3.1.3.2 SHAFT REPAIR ACCIDENT LEADING TO WATER INFLOW

This may be caused by:

a) having to replace part of the shaft lining due to degradation of the material,

b) having to drill into the shaft lining to install or replace brackets for pipes and cables in the shaft,

c) drilling through the shaft lining for grouting purposes.

In each of these instances, the safety of the particular operation will be dependent on the strength of the shaft lining material, and the presence or absence of water under pressure behind the shaft lining.

The flooding of the Cominco Potash mine in 1970 was the direct result of an accident during shaft repair operations. During routine grouting of the concrete lining, a drill hole intersected a major source of water that was under high pressure (Prugger, 1979). The primary cause of the accident was the intersection of the unexpected water under high pressure.

3.1.3.3 SEISMICITY INDUCED BREACHING OF THE SHAFT LINING

Breaching of the shaft liners can, in principle, be caused by natural occurring earthquakes or by mining induced seismicity. Earthquakes are considered to be a rare event in Saskatchewan. The largest known event, estimated magnitude of 5.5 on the Richter Scale, occurred in 1909, near the U.S. - Canada Border. From November 18, 1979 to May 11, 1982, six seismic events, ranging in intensity from 2.4 to 3.0 on the Richter Scale, have occurred in the vicinity of the Cory mine near Saskatoon. Subsequent studies initiated by PCS Mining have shown that the seismic activity was induced by the mining operation. For additional details, the reader is referred to Gendzwill (1981) and the Mining Induced Seismicity section of this report.

Preliminary studies by PCS Mining have shown that the <u>designed strength</u> of the shafts is adequate to withstand earth tremors of maximum intensities in the order of 3.0 (Richter Scale) expected from mining induced seismicity, and can withstand an earthquake of magnitude 5.5 on the Richter Scale.

The concern of the mines is that the designed (and constructed) strength of the shaft liners may be weakened by corrosive chemical action, over time, of the formation waters in contact with, and seeping through, the shaft lining (as discussed below). Such activity could conceivably produce a weak section in the shaft lining which could fail due to an earth tremor or earthquake.

3.1.3.4 SHAFT MATERIALS FAILURE LEADING TO WATER INFLOW

This scenario assumes that the strength of a section of the shaft liner has been degraded by the corrosive action of formation waters in contact with or seeping through the shaft liner, and that the bulk seepage does not relieve the pressure of the formation water. At some point in time, the built-up water pressure would exceed the degraded strength of the liner, and the liner would then fail.

Water seepages have developed in shaft linings. In most instances, these occur at the joints in the concrete, or at the joints between the tubbing rings. Such seepages are capable of being rectified by grouting, tightening of the sealing rings or other such repair procedures. What cannot be determined at this time, due to the lack of instruments and techniques, is the degree and amount of corrosive degradation of the shaft liner.

3.1.3.5 SHAFT FAILURE DUE TO BUILD-UP OF COMPRESSIVE FORCES AGAINST THE SHAFT LINING

This would occur by the transfer of pressures preferentially to the shaft liner, particularly in an area of partially degraded liner, causing hoop stress to exceed the shaft liner strength, with resultant rupturing of the liner. Such transfer of pressure could conceivably be caused by:

a) differential subsidence in the shaft pillar,

b) by salt "flowage" in the salt sections above the mining level, or

c) by dissolution of the rock or salt behind the shaft lining.

No rock mechanics measurements have been taken in any of the shafts of the participating mines to determine if any build-up of rock pressures, particularly creep of the evaporite formations, is occurring behind the shaft lining. The difficulty is the restriction imposed of not penetrating the shaft lining in order to take measurements. The concern is that such penetration of the lining would introduce a risk factor that would be potentially much greater than the risk of a build-up of compressive forces. Even with this restriction, it appears that no measurements have been made inside the shaft to determine if the shaft lining is deforming by differential closure of the shaft walls. Such movements may be detected by the change in diameter of the cross-section of the shaft at different places in the shaft.

3.1.4 THE JUSTIFICATION FOR RESEARCH AND DEVELOPMENT OF INSTRUMENTATION TO MEASURE SHAFT STABILITY

In each of the five areas of concern described above, two factors are common to all, and must be present to cause, or contribute directly to, the possible failure of the shaft lining and the possible consequent flooding of the mine. These two common factors are:

a) the chemical corrosion and weakening of the shaft liner from the action of formation waters in contact with the liner, and

b) the presence of formation water under pressure at or near the breach.

The problem is the lack of suitable, non-destructive methods or procedures to physically test the structural strength and integrity of the shaft lining; to determine the location, degree and extent of corrosive degradation of the shaft linings; and to identify the presence or absence of formation waters behind the linings.

3.1.4.1 PRESENT INSPECTION PROCEDURES

Saskatchewan mining regulations require that each shaft is inspected at least once a week. Although the regulations are primarily oriented to the safe transportation of men in the shaft, the shaft inspection does include the shaft lining.

These inspections are visual only. Particular attention is paid to rusting of the cast iron or steel tubbing, the spalling of the concrete lining, and the location and rate of water seepage into the shafts.

3.1.4.2 PROHIBITION OF DESTRUCTIVE PHYSICAL AND CHEMICAL TESTING OF SHAFT LININGS

The only known method of physically testing the shaft lining materials requires "coring" of the lining or other similar "destructive" sampling. Such destructive procedures are not acceptable to the industry, out of the risk of flooding such as occurred at Cominco's operation. It is therefore impossible to determine the degree of

deterioration of the structural strength and integrity of the shaft lining with present technology. Because of the same prohibition, it is not possible to determine what is happening behind the shaft lining.

3.1.4.3 RESEARCH AND DEVELOPMENT REQUIREMENTS

There is a requirement for the research and development of instrumentation and techniques to:

a) locate and measure the degree and extent of corrosive degradation of the shaft lining,

b) locate formation water behind the shaft lining

c) detect and measure build-up of compressive forces acting on the shaft lining.

The research and development carries the caveat that the techniques and instrumentation cannot physically penetrate into or through the shaft lining.

3.2 MINING INDUCED SEISMICITY

Induced seismicity is a new phenomenon in the potash mining area of Saskatchewan. On November 18, 1979, an earth tremor measuring 2.4 on the Richter Scale occurred in the vicinity of the Cory potash mine west of Saskatoon. Subsequent to that event, five additional tremors, the strongest measuring 3.0 (Richter Scale), have been identified as having

occurred at or near the Cory mine. Investigations initiated by PCS Mining, utilizing the staff and facilities of the Geological Sciences Department, University of Saskatchewan, have proved beyond reasonable doubt that the earth tremors occurring at or near the Cory mine are mining induced seismic events.

On February 10, 1984, an earth tremor measuring 2.4 (Richter Scale) was recorded as having occurred near the vicinity of the Central Canada Potash mine east of Saskatoon. In the Esterhazy area of Saskatchewan, seven earthquakes have occurred since November 1976 (Horner, 1983).

Gendzwill of the University of Saskatchewan has been the principal investigator of mining induced seismicity on behalf of PCS Mining. Several papers have been published detailing the results of these investigations, including possible mechanisms for the cause of the earth tremors [Gendzwill, D.J. et al, (1982); Horner, R.B., (1983) Prugger, A.F. and Gendzwill, D.J., (1983); Gendzwill, D.J., (1983).] The reader is referred to these papers for the pertinent details.

3.2.1 CONCERN REGARDING MINING INDUCED SEISMICITY

The occurrence of induced seismicity associated with potash mining has raised several areas of concern to the mines:

1. The safety of the underground workers, which includes the integrity of the mine.

2. The safety and integrity of mining and milling installations - shafts, brine ponds, brine injection wells and abandoned exploration wells, surface facilities.

3. Maximum protection of the public residing near the mine.

In order to alleviate these concerns, it is necessary that the cause of the earth tremors, the mechanism, and the maximum size of event that may occur, must be better understood.

3.2.2 PROBABLE CAUSATIVE FACTORS IN MINING INDUCED SEISMICITY

The earth tremors that have occurred at the Cory mine are the only series of potash mining induced seismic events studied to date. This is, in part, due to the fortuitous location of the mine within reasonable proximity to the University of Saskatchewan in Saskatoon.

Gendzwill (1983) has determined that all of the earth tremors at Cory have occurred at a shallow depth, in the order of a kilometre. One specific earth tremor was interpreted as occurring at a depth of 960 m.

As the mining level is at or just below 1 000 m from surface, and no visible damage, rock bursts or measurable movement of the floor or back has occurred, it is most probable that the epicentres of the earth tremors are above the mining horizon. This probability leads directly to the suspicion that the cause of the earth tremors is directly related to:

a) subsidence

b) mining pattern

c) rate of extractions, and

d) extraction ratio

It follows that these areas require research to determine the cause and mechanisms of mining induced seismicity in order to minimize or eliminate the phenomenon.

3.2.3 MINING PATTERNS

The mining pattern utilized at each mine has evolved over the years through primarily trial and error, with the application of rock mechanics measurements and principles. Particular geological conditions have led to the use of a particular pattern of mining.

In the Saskatoon area, the presence of clay seams in the roof led to the use of the chevron system of stress-relief mining to control roof fall.

In the Esterhazy area of Saskatchewan, the lack of clay seams in the roof has permitted the use of a different mining pattern - long room and pillar, as at Rocanville.

The long room and pillar pattern should be capable of spreading the stresses developed over the mine in a more uniform manner than the chevron style pattern.

3.2.4 EXTRACTION RATIO

The extraction ratio is determined by the mining pattern. The long room and pillar pattern used in the Esterhazy area should allow for a more uniform extraction ratio (35 - 45%) spread over a larger area than the chevron system used in the Saskatoon area.

The chevron pattern, which utilizes yield pillars in the chevron panel and abutment pillars between panels, does create a concentrated high extraction ratio in the panel, although the overall extraction ratio of a number of such panels in one area would be lower (60% and 35 - 40% respectively, on average).

It may be that localized areas of non-uniform stresses build-up over chevron pattern mining areas because of the higher extraction ratio in the chevron panels.

3.2.5 RATE OF EXTRACTION

At the time of the first earth tremor at the Cory mine, three mining machines were operating in the same general area of the mine (in adjacent panels). This meant that the rate of extraction was higher than the normal for that particular area. This probably increased the rate at which stresses built-up over the mine in that particular area.

3.2.6 SUBSIDENCE

The accepted practice in potash mining is to insure a gradual and uniform subsidence of the overburden to prevent the build-up of stresses in the overlying formations. Mining design, mining pattern, extraction ratio and rate of extraction are factors that influence the subsidence.

Over the years, the mines have modified all of these factors as they have gained experience from ground reactions to the various different mining patterns and related extraction ratios used in their operations. No direct analytical relationship has been established as yet between these factors and the subsidence. This is due, in part, to the relative slow rate of subsidence in the order of 10's of millimetres per year; the accuracy of the surface surveys over the large area of the mine (100's of hectares) relative to the overall magnitude of ground movement; and the rock mechanics measurement programs underground which vary in complexity at the different mines.

3.3 GEOLOGICAL ANOMALIES

The potash deposits of the Prairie Evaporite Formations in Saskatchewan are essentially considered to be flat-lying and continuous within the mining properties. Geologically anomalous structures, which have been encountered at all Saskatchewan potash mines, vary from relatively small barren zones and channel-like features within the ore zone, through to major collapses of the complete evaporite and the limestone members above (Mackintosh, 1983).

All other anomalous structures encountered have been the result of the action of water dissolving the salt and/or potash during deposition of the Prairie Evaporite Formation, prior to the consolidation of the Formation, or post-consolidation following the deposition of some or all of the overlying formations.

Although the occurrences of geological anomalies affect ore grade and may result in the isolation of, or the abandonment of, large areas of potential ore, the primary concern is that these anomalous structures may pose, or do pose, a hazard to the mining operation.

The primary concern of the mining operators is that these anomalous structures may contain water, or may be, or develop into, conduits through which water may be introduced into the mining horizon from the underlying and overlying water-bearing formations.

3.3.1 TYPES OF GEOLOGICAL ANOMALIES

Mackintosh (1983) has identified and described three main types of anomalous structures that occur at the Cominco Potash mine west of Saskatoon. Similar features have been recognized at most of the other potash producers. These types are incorporated and expanded upon as required in the following descriptions:

3.3.1.1 WASHOUTS

Washouts are salt-filled structures which transect the normal bedded sequence of the ore horizon and disrupt or obliterate the stratigraphy. Although commonly V- or U- shaped, they also occur as dike-like or vein-like concentrations of salt that cut across the normal horizontal bedding.

The two characteristics that distinguish washouts from collapse structures (described below) are that the contact of the washout with the adjacent bedding is generally sharp and is cross-cutting the bedding, and the adjacent bedding is essentially not disturbed.

The dimensions of washouts are variable from less than a metre in width to several tens of metres. The lateral dimension is also variable, but most appear to be quite short. Due to the geometry of the mining patterns utilized in the operations, - room and pillars on one level, heights of rooms varying from 2.4 to 4.9 m, - vertical and lateral dimensions are often indeterminate.

The typical washout will contain clear to translucent, coarse-grained salt in the upper 2/3 or greater portion of the structure, with the bottom portion often filled with a concentration of insoluble material.

The consensus of opinion is that washouts formed as the result of erosion of the salt at or near the surface during periods of exposure or near exposure. Brine flowing to adjacent sink holes in the salt is

considered to develop sufficient energy to cut stream channels in the layers of insoluble clay particles and potash crystals. A few washouts have developed below the surface as indicated by the presence of slabs of overlying salt beds in the material filling the channel.

The channel itself is considered to fill as a result of the eventual plugging or filling of the sink hole with insoluble material and salt, the resultant loss of energy flow terminating the stream, and the deposition of salt from stagnant brine in the inactive channel.

3.3.1.2 LEACHED ZONES

Leached zones are characterized by the partial or complete absence of potash in an otherwise normal, continuous and structurally undisturbed stratigraphic sequence. Shale beds are continuous through the zone, but the ore beds have been partially or totally replaced by salt, or have been removed with resultant thinning of the section.

As described by Mackintosh (1983), . . .

> "An overall colour change from the normal rich salmon-orange to a dark glassy gray-brown is the most obvious indication of a leached zone along with a change in colour of the shale beds from their normal greenish-gray to a rusty-brown. The boundary between leached and unleached material commonly is sharp, in places showing potash enrichment immediately below

the contact. Shale beds commonly appear to have protected lower beds from leaching solutions. Leached zones generally are circular or oval in plan, and saucer shaped in section. They range in diameter from a few metres to as much as 400 m but a few are linear, being up to 20 m wide and greater than 1 600 m long. They occur randomly throughout the mine but can occur on the edge of complex collapse structures. Because the normal bedding is so well preserved, the zones must develop as the result of the low energy percolation of water that is nearly saturated with respect to sodium but undersaturated with respect to potassium, This situation could develop as the result of a local depression in the water table during a dry cycle in the depositional sequence with water, freshened by rainfall, moving downward along narrow fractures or between the edges of polygonal structures, or through other zones of higher permeability."

3.3.1.3 COLLAPSE STRUCTURES

Collapse structures are the least understood and therefore probably the most controversial geological phenomena encountered in potash mining. The typical collapse structure is described as a U or V shaped structure filled with brecciated, recemented or recrystallized material in which the strata is no longer recognizable. In most instances the strata adjacent to the collapse structure dips at a distinct angle to the horizontal, suggesting a down-warping, removal, or collapse of the beds

below the observed collapse structure. In some cases, as described by Mackintosh (1983), . . . "broken fragments of bedded strata with recognizable marker beds have been observed, folded, overturned and out of sequence within the structure".

In some instances, blocks and fragments of overlying Second Red Bed and Dawson Bay Formation have been recognized in the breccia contained in the collapse structure. Allan, Cory and Lanigan mines have experienced such occurrences in collapse structures.

The size and shape of collapse structures in the Prairie Evaporite Formation can vary from cone or disk shaped structures tens of metres in diameter to major collapse features measured thousands of metres in width and upwards of 12 000 m long. The Saskatoon Solution Collapse Structure and the Colonsay Collapse Structure are examples of the latter.

Discrete troughs have been intersected at Lanigan. Although the bedding has been "down-warped" for several metres over a width of tens of metres, the bedding is still intact with no evidence of brecciation. These are considered to be either the edges of a localized collapse structure or one that was only partially developed.

The origin and mechanism of formation of collapse structures is somewhat controversial. Gendzwill (1978) describes a process for the initiation of collapse structures by dissolution of the base of the salt by Winnipegosis formation water moving along the surface of Winnipegosis

reefal mounds. Bediz (1983) has postulated that salt flowage down the flanks of Winnipegosis mounds may have been the mechanism for the development of some collapse structures, particularly the smaller sized anomalies intersected in the mining horizon. Other authors (for example see Holter, 1969, p. 54 - 56) have postulated that tectonic activity, particularly basement movement, or pre-Prairie Evaporite structural anomalies may have been responsible for the formation of the large-scale collapse structures such as the Saskatoon Solution Collapse Structure. The possibility that at least some of the smaller anomalies may have originated from above the Prairie Evaporite should not be ignored.

3.3.1.4 WINNIPEGOSIS MOUNDS

The Winnipegosis Formation, a dolomitic limestone, occurs at the base of the Prairie Evaporite Formation throughout the area containing the potash deposits. In the Saskatoon area, excluding the Lanigan Mine, irregularly shaped mounds or carbonate banks, sometimes referred to as reefs, have been identified by seismic survey and drilling. As these carbonate build-ups have not been positively identified as "reefs", the term "mounds" is the preferred descriptive name used to describe these features.

In the Saskatoon area, these mounds attain thicknesses on the order of 90 to 110 m, and have proven to be the loci of "problem mining conditions". Much thinner mounds, of the order of 30 m thick, have been indicated by seismic to occur in the Lanigan and Esterhazy areas.

These thinner mounds do not appear to have any effect on the mining conditions experienced in the potash horizon.

In the Saskatoon area, the role of the thicker Winnipegosis mounds in the development of geological anomalies in the mining horizon is not completely understood. Since the mid 1970's, surface seismic surveys have become a routine technique used by the mines to explore future mining areas. These surveys have revealed that, in places, there is a distinct thinning and development of depressions or "lows" at the top of the Prairie Evaporite Formation over some of the Winnipegosis mounds. Underground mining experience is showing that collapse structures or other geologically anomalous features often occur at, or adjacent to, the edges of these mounds or Winnipegosis "highs". Conversely, where the Prairie Evaporite Formation is indicated to be a broad, relatively flat area, whether over Winnipegosis highs or lows, mining conditions are generally found to be good, with few, if any, anomalies in the mining horizon.

In the case of the Lanigan mine, no significant reefal mounds (i.e. 100 m or higher) have been identified as underlying the property. However, numerous small collapse structures and troughs have been intersected in the mining horizon. Several large collapse structures or depressions, measuring in the hundreds to several thousands of metres in long dimension have been identified by seismic and drilling within or at the edge of the mining property.

At Rocanville, there is a notable absence of collapse structures in the mining horizon. As at Lanigan, only a few low (30 m) mounds have been observed on the top of the Winnipegosis seismic maps. However, at least two rather large washouts or leached structures over 100 m in diameter, have been intersected in the mining level. These washouts contained water dated by oxygen isotope ratio to be Dawson Bay and lower Souris River, and visible vugs or small cavities were noted in the coarse grained salt. The relationship to the Winnipegosis is not clear at this time.

In summary, collapse structures in the Saskatoon area appear to be somewhat related to Winnipegosis mounds. This relationship is not obvious at Lanigan. At Rocanville, the presence of formation waters from <u>above</u> the Prairie Evaporite suggests a process that originated at or above the top of the salt.

3.4 WATER INFLOWS

The safety of the workers underground is the first concern of any mine. In the potash mining industry, the second concern is the possibility of the mine being flooded by formation waters present above and below the mining horizon. To a lesser extent, there is the concern of mining into connate water or brine in the potash horizon. However, as the areal source of connate brine would be limited, the risk imposed by such an accident is considered much less than that posed by formation waters, but, it is still a risk that cannot be ignored.

Three possible sources of water inflows, - the shafts, abandoned exploration wells and brine injection wells, are treated as separate problems elsewhere in this report.

3.4.1 THE POTENTIAL RISK FROM FORMATION WATERS

The mineable potash beds in Saskatchewan that are exploited by conventional underground mining techniques occur at approximately 1 000 m below surface. Between the surface and the potash bed, eleven or more known water bearing horizons are present over every mine.

The Blairmore Formation (now known as the Mansville Group) is perhaps the most familiar water-bearing formation primarily because of the problems it posed to earlier attempts at potash shaft sinking. This formation is encountered at depths around 380 m below the surface. It is essentially an unconsolidated to poorly consolidated water-bearing sand, silt, mudstone and shale horizon, ranging in thickness from 60 to 150 m, and with water pressures ranging from 2 700 to more than 5 500 kPa.

Between the Blairmore and the Prairie Evaporite Formation, which contains the potash beds, five to ten or more water-bearing horizons, pressurized up to more than 7 500 kPa, are present over the mining horizon at every mine.

Below the Prairie Evaporite Formation, the reef-forming, water-bearing Winnipegosis Formation is considered to be a possible source of water that could pose a risk to the mining operation. However, as the mining horizon is underlain by a significant thickness of salt and anhydrite, in excess of 50 to 100 m, the threat to the mines from this formation is considered to be via geological anomalies providing conduits for the Winnipegosis formation waters. Geological anomalies also pose a similar threat from the overlying water-bearing formations. Because of their importance, these anomalies are discussed under a separate section in this report.

3.4.2 THE THREAT FROM OVERLYING WATER-BEARING FORMATIONS

The over-riding concern here is that the mining system employed causes fracturing into the roof and the formations above the potash level, thus allowing water to enter the mining horizon.

The first defence against such an occurrence is the salt back above the mining level. This salt cover ranges in average thickness over the entire potash mining area from 24 feet (7.3 m) to over 90 feet (27.4 m), but is known to vary over all of the mines. In addition to its variable thickness, it is known that the salt cover has been breached in past geological time. For example, Cory Division mined into a slab of limestone in a disturbed area at the mining level. Elsewhere, such as at Lanigan Division, collapse breccia has been intersected at the mining level. Thus, the seal provided by the salt-back cannot be considered totally reliable.

The salt cover is overlain by the Second Red Bed which is essentially a reddish-brown dolomitic mudstone, very poorly consolidated, with local greenish bands (Dunn, 1982). It forms the lowest member of the Dawson Bay Formation which overlies the Prairie Evaporite Formation. In the Saskatoon area, which is considered to be that area encompassing six of the potash mines (Cominco, Cory, PCA, Allan, Central Canada, Lanigan) the Dawson Bay is considered to be "dry". In the Esterhazy-Rocanville area, the location of IMC and Rocanville Division mines, the Dawson Bay is considered to be "wet".

The term "dry" is somewhat misleading in that localized "pockets" of water do occur in the Dawson Bay Formation over the Saskatoon area mines. Although it has been postulated that these brines represent localized pockets of connate water, oxygen isotope studies by the University of Saskatchewan have shown that younger formation water is present in some of these brines. This indicates a connection between waters of the Dawson Bay Formation and that of the overlying formations.

Regional hydrogeological studies by Sproule Associates Limited (1980, 1981), carried out on behalf of all of the conventional underground potash operators in Saskatchewan, have indicated that the various formation waters are interconnected, at least on a regional basis.

The oxygen isotope evidence and the Sproule study shows that, in theory, any water entering the mine workings may originate from a localized pocket in the Dawson Bay Formation, from connate brine in the Prairie Evaporite Formation, or from water-bearing formations above the Dawson Bay Formation.

3.4.3 MINE DESIGN AS A FACTOR IN PREVENTING WATER INFLOWS

The basic design and layout of a potash mine is a main factor in protection from formation water inflows. The major design objective is to optimize ore extraction, to prevent back failure, and to avoid sudden stress relief with consequent fracturing of the overburden or the collapse of the overburden.

Over the years, the mines have utilized rock mechanics programs combined with surface subsidence observations to monitor the reaction of the roof and overburden to different mine layouts and geometry of room and pillar widths and/or heights, and extraction ratios. Water inflows have occurred despite the precautions observed by the mines in their operations.

The Cory mine, for example, has experienced four water inflows in the past fourteen years that have proven to be more of an operational irritant rather than a major hazard. Chemical and oxygen isotope analyses indicate that the waters originated in the Dawson Bay Formation. The initial rate of inflow was of 5 gallons per minute or less (0.02 m^3/min) which gradually decreases over a period of several weeks to a month to a steady drip. Over time, it has been observed that these rates increase for a period of time and then decrease. This fluctuation or pulsation has been attributed to a salt plugging and then dissolutioning of the conduits or channels. An alternative theory is that the gradual subsidence of the overburden closes off the channel ways. The water then leaches out new channels which in turn are closed

off by on-going subsidence. The pulsation and the fact that the water is of Dawson Bay age suggests that the source is an isolated or localized pocket of formation water in the Dawson Bay.

In the case of the Cory mine leak, the fact that the inflows occurred some time after the entries had been excavated clearly suggests that the inflows were related to the closure of the overburden above the mining level. This is substantiated to a degree by the fact that the inflows originated at holes drilled in the roof, and that on one occasion, the rock mechanics staff noted an increase in the rate of roof closure immediately prior to the water inflow. The consensus of opinion is that differential "sagging" of the overburden produced fracturing or conduits for the Dawson Bay water to migrate down to the mining level.

At the Lanigan mine, three water inflows have occurred in the past four years. One of these inflows was located in a geological anomaly. The other two inflows were in areas that were not obviously "disturbed". Oxygen isotope analyses have detected the presence of both Dawson Bay and the younger Lower Souris River formation waters in the two inflows, clearly indicating a connection between the two formations.

At the Lanigan mine, the water inflow was observed at the mining level a few hours after the entry had been cut by the mining machine. In this case, oxygen isotope analyses indicated the presence of both Dawson Bay and younger Lower Souris River Formation waters. In this instance, it must be assumed that the water inflow originated from a pre-existing presence of both waters at the mining level, and that the machine cut

into (or very close to) the "pocket" containing the already mixed waters. The time factor is too short to invoke induced fracturing from the mining level into and through the Dawson Bay to the Lower Souris River Formation as the contributing cause of the flow.

The significant water inflow at the Central Canada Potash mine may be caused by mining induced fracturing of the overburden (or roof) into water-bearing Dawson Bay in an anomalous thin salt-back area (Tofani, 1983) in Panel 209.

The experiences of Cory, Lanigan and Central Canada Potash mines illustrate the need for the research and development of instruments and techniques to measure the salt-back thickness ahead of and above the mining level and the presence of water either ahead of or above the mining horizon.

3.5 BRINE INJECTION WELLS

The potash industry in Saskatchewan began to use brine injection wells in 1968. The original concept was that brine ponds would contain the excess brine used in the milling processes, and that evaporation would reduce the level in the ponds. It became necessary to develop a means of disposing of the brine to reduce the quantity stored on surface; brine disposal wells were the obvious solution.

3.5.1 INSTALLATION OF BRINE INJECTION WELLS

Brine injection wells are basically oil wells operating in reverse. Although the actual installations are more complex, the following is a generalized description of a typical brine injection well.

The well is drilled using oil well drilling rigs and technology to intersect a favourable, porous and permeable formation, generally a sandstone. The well is then cased to that formation and the piping is perforated at the porous horizon. Packers or seals are installed above and below the perforated section. The excess brine is then pumped from surface to the perforated section and into the porous formation.

All of the participating potash mines inject the brine into the same formations, Winnipegosis and/or Deadwood, at 4 500 to 5 500 feet (1 372 to 1 676 m) below the surface. As the Saskatchewan mines are around 3 000 feet (914 m) below surface, the disposal formation is around 1 500 - 2 500 feet (450 - 750 m) below the mining horizons.

Four of the five mines have installed the injection wells in the shaft pillars, and four of the five mines have the injection wells within 600 to 700 feet (183 to 213 m) of mine workings.

The actual utilization of the injection wells tends to vary due to the season of the year (least in summer because of the evaporation from the brine ponds) and the size of the brine ponds on the surface. The usual capacity of each well is from 500 to 1 000 gallons per minute (2.3 to 4.5 m^3/min).

3.5.2 MINING AND ENVIRONMENTAL CONCERNS REGARDING BRINE INJECTION WELLS

Concern has been expressed by all the participating PCS Mining mines in this project that there may be a degree of mine instability being affected by the injection of brine into the formations below the mining horizon. At the same time, the Plant Engineering staff at the same mines expressed the strong desire to inject greater quantities of brine than at present. This same desire was expressed by the Environmental Affairs department of PCS Mining. The concern of the plants and the Environmental Affairs department is to reduce the environmental risks associated with large brine ponds on surface. Although the environmental aspects of brine injection do not directly relate to mine stability, it was the opinion of the participants that it would have to be addressed as part of the mine stability study.

3.5.3 THE PROBLEM OF BRINE INJECTION WELLS - CONCERN FOR MINE STABILITY

The most significant concern to mine stability is the lack of knowledge as to what is happening to the injected brine. There are several unanswered questions:

a) Does the injection of the brine result in a build-up of the pressures in the disposal formation? If so, could the brine injection result in hydrofacturing of the formations below the mining horizon?

b) Could the injected brine penetrate into, lubricate, and reactivate old faults in the formations below the mining horizon?

c) Could the injected brine in time dissolve the salt from the salt-healed fractures and eventually penetrate up to the mining horizon and cause flooding of the mine?

d) If the disposal formation is under pressure from the brine injection, and the well casing failed due to subsidence, seismic activity or materials failure, could the brine and/or formation waters migrate into and cause flooding of the mines?

3.5.4 CONCERN FOR ENVIRONMENTAL EFFECTS

The environmental and mill or plant staffs have two basic concerns that result from the storage of brine in large ponds on the surface:

a) Although most brine ponds have a plastic or earthen liner, seepage of brine does occur, with resultant contamination of adjacent land, water and groundwater formations.

b) Brine ponds are retained by earthen dikes. The concern is that seepage can occur through the dike, and the possibility that the dike is breached, resulting in surface contamination of a large area.

The plant staffs and Environmental Affairs wish to reduce the amount of brine that is stored on surface at the present time. For the future, when the plants are expanded, the desire is that the quantity of brine stored on surface is not increased. To meet this, the number of brine injection wells would have to be increased.

3.5.5 MONITORING OF INJECTION WELLS

Government regulations require that a pressure test of the injection well is carried out on an annual basis. It is uncertain that any of the mines monitor the pressure on the well over any significant length of time.

Although the mines have indicated that the quantity of brine injected is recorded and available on a daily/weekly/monthly basis, however, in the majority of cases, it does appear from conversations with the plant staff and others that the quantities recorded have been questioned, at least in the past.

The chemical composition of the brine is monitored during the milling cycle, but the actual composition of the injected brine may vary due to the addition of fresh water used at times to flush the injection pipe to improve the rate of injection.

It is suggested that the data base for studies of pressures of injection and chemical composition requires additional studies.

3.6 POTASH EXPLORATION WELLS

Each of the mine properties has been explored by exploration wells that intersected or penetrated through the mining levels. These wells are presumed to be sealed by cement plugs. The concern is that the mining operation may intersect, or mine close to, the abandoned exploration wells, and that the wells have not been adequately sealed, or that the plugs are no longer functioning. In view of the fact that each well penetrated through water-breaing formations above the mining horizon, the potential for water entering the mine via the exploration wells cannot be disregarded.

3.6.1 DRILLING AND ABANDONMENT PRACTICES

Potash exploration wells drilled in Saskatchewan use standard oil well drilling technology with the exception that saturated brine or an oil based drilling fluid is used during coring of the salt and potash sections of interest. The drilling and abandonment procedures are outlined in the regulations issued by the Province under the Oil and Gas Conservation Act. These regulations require that any well drilled into or through the Prairie Evaporite shall be plugged with a brine saturated cement to 500 feet (152.4 m) above the top of the Prairie Evaporite, and that each porous or water-breaing zone above this plug is isolated by another cement plug. The present practice is to <u>completely</u> plug the exploration wells with cement.

A report describing the abandonment program is submitted to the Saskatchewan Department of Energy and Mines at the conclusion of the drilling. These reports are on permanent file with the Department, and are available to anyone interested. However, there is no way of determining from the report that the cement plug did, in fact, seal off the water bearing horizons and if the seal is still functioning at this time.

3.6.2 RELIABILITY OF CEMENT PLUGS

Available statistics on borehole degradation from the Michigan salt basin in the United States (Pinaire and Snider, 1973) indicate that in 40 year old wells, there is an overall probability of 25% that the plugs are either missing or soft. This estimate is based on 20 drill holes containing 49 plugs which were rebored after 40 years.

It was found that:

- 50% of the bottom plugs were either soft or missing,
- 10% of top plugs were missing, and
- Overall, 22% of all plugs were missing and 12% were porous.

Obviously, there must be qualifiers put on the age of the drill holes, plugging methods, quality control, depth of the plug, composition of the plug and the environment down the hole. Most of the wells drilled in the Saskatchewan potash mining areas are in the order of 20 years old. It is therefore possible that some of the plugs are no longer functioning as proper seals.

3.6.3 SAFETY PRECAUTIONS UTILIZED BY THE OPERATORS

The present procedures followed by the mine staff to alleviate the possible hazards posed by the presence of abandoned exploration wells in the mining area is to leave a pillar, 500 feet to 900 feet in diameter (152.4 m to 274.3 m), based on the surface location of the exploration well. To date, no mine has intersected an exploration well in their underground workings.

Although the leaving of a safety pillar around the surface location of an exploration well is a reasonable precaution, it should be noted that not all exploration wells, upon completion, were subjected to a directional survey to determine the location of the bottom of the hole.

Two of the mines have not accurately determined, on the surface, the exact location of the exploration wells in their lease areas. Because the drilling was done as far back as the late 1950's, it is now impossible to do this from the actual well hole. They must now rely on the surveyed location plan made prior to the drilling of the well. This is not always reliable, as the hole may have been moved several tens of feet to allow for a better rig set-up, and the new location not re-surveyed.

3.6.4 POSSIBLE HAZARDS POSED BY ABANDONED WELLS

The most obvious hazard is the mining into an abandoned exploration well in which the cement plug has degraded and no longer seals the hole

from formation waters. A similar hazard exists if the mine driveage is sufficiently close to the hole and the normal closure of the mine workings creates factures or conduits allowing the water into the mine from the well in which the plugs are no longer functioning.

There is a possible hazard from any well in which the plugs have degraded and no longer seal off the formation waters. There is a similar possibility that a properly plugged well whose seals are functioning could become a potential hazard due to the effects of subsidence, mining induced seismicity, or natural earthquakes. In both of these possibilities, there is the concern that formation water above the mining level could eventually work its way down the well and by solutioning or following mining induced fracturing, eventually end up in the mine.

Clearly, abandoned exploration wells cannot be ignored as potential hazards.

4. RECOMMENDATIONS FOR RESEARCH AND DEVELOPMENT TO ALLEVIATE OR ELIMINATE THE PROBLEMS AFFECTING POTASH MINE STABILITY

The recommendations described in this section are the result of discussions with the various mine staffs and the expert consultants used by the industry in the fields of geology, geophysics and rock mechanics, as well as the experience gained by the industry in attempting to resolve the problems encountered during mining operations.

4.1 RECOMMENDATIONS FOR RESEARCH AND DEVELOPMENT CONCERNING SHAFT STABILITY

The three major factors affecting shaft stability have been identified by the mines as:-

1) deterioration or degradation by the corrosive action of formation waters on the concrete and tubbing that make up the shaft linings,

2) the build-up of differential pressures on the shaft lining, and

3) the presence of undetected water behind the shaft lining.

From a review of the available literature, and from discussions with the various mine staffs, it has been determined that there are no known suitable (non-destructive) instruments available at this time to monitor or detect the magnitude or extent that these factors may be affecting the stability of the shafts. The available technology incorporates the drilling or coring of the shaft lining for purposes of detection or measurements. However, the restriction imposed by the operators that any instruments, methods or procedures can NOT physically penetrate or destroy the shaft lining, rules out the use of available instrumentation.

New instrumentation, methods or procedures must be researched or developed to enable the mine staffs to determine the effect of each of the above listed factors on the stability of the shafts.

4.1.1 INSTRUMENTS AND TECHNIQUES REQUIRED TO MEASURE DEGRADATION OF SHAFT LINING MATERIAL

It is of paramount importance that a non-destructive method be researched and developed to determine if the shaft lining has deteriorated due to the corrosive chemical action of the formation waters on the material utilized in the shaft lining. Such a method must be capable of detecting and measuring the degree and extent of the corrosion without physically penetrating the shaft lining itself.

The design of the lining and the strength of the materials used in construction of the lining have taken into account the expected pressures

and chemistry of the environment in which the shafts are installed. Despite these precautions, water seepages, rusting of the tubbing and spalling of the concrete does occur, as reported by the mines. These visual observations made from inside the shaft lead directly to the concern that similar activity may be occurring on the other side of the lining, and that the critical strength and/or thickness may deteriorate and ultimately lead to failure of the shaft lining.

It is recommended that a research program be initiated to develop non-destructive instruments and procedures to detect and measure the deterioration of the material of the shaft linings due to the corrosive chemical action of the formation waters. Such a research and development program should include:-

1) A world-wide survey to determine what, if any, techniques are used in other countries to detect and measure the deterioration of shaft lining material, and to document shaft failure due to such deterioration.

2) Sonic, ultrasonic, radar and other similar technologies should be investigated to determine their suitability for development as methods to detect and measure the deterioration of the shaft lining material.

3) Laboratory testing of the lining material to determine the rate and degree of deterioration of the strength of the lining materials over time in the particular environment of potash mining.

Perhaps the most important recommendation is to have a multi-disciplinary group of mine engineers, shaft specialists, chemists, metallurgists and similar experts address and recommend additional methods or technologies for research and development to solve the problem of deterioration of the shaft lining material.

4.1.2 INSTRUMENTS AND TECHNIQUES REQUIRED TO DETECT AND MEASURE BUILD-UP OF ROCK PRESSURES ON SHAFT LININGS

Instrumentation is available to measure the distortion or the deforming of the shaft lining from its original circular shape by measuring for the differential closure of the shaft wall. Unfortunately, no measuring stations were established during shaft construction at any of the mines. There is no record of any measurements made since the shafts were placed in operation. There is a total lack of a data base upon which to establish criteria for interpreting any movement detected by available instrumentation.

For reasons stated earlier, mine operators will not permit the physical penetration of the shaft liner to determine if the rock formations behind the liner are moving (closing), or cavity formation due to solutioning. It is equally unacceptable to drill holes into or partially through the shaft liner to install stations to measure movement. The concern here is that such a drill hole would intersect an unknown material weakness in the concrete or tubbing and possibly lead to lining failure, or that the drill hole itself becomes the locus of degradation by corrosion.

It is recommended that non-destructive instrumentation and techniques be researched and developed to allow the operators to determine if there is a build-up of rock pressures behind the shaft lining, and to measure and interpret any such build-ups.

It is difficult to recommend any specific line of research to obtain the required results that does not involve physical penetration of the shaft lining in order to install sensors. Radar, sonic, ultrasonic and, perhaps, laser technologies should be investigated to determine their applicability to the problem.

4.1.3 INSTRUMENTS AND TECHNIQUES REQUIRED TO DETERMINE THE PRESENCE OF WATER AT OR NEAR THE SHAFT LINING

It is important that such instrumentation be developed to prove the presence <u>or</u> absence of water behind any area of the concrete portions of the shaft lining. This knowledge will be required if repair or replacement of the concrete lining is required at any future time. It is especially important to determine if formation waters are ducting behind the shaft lining to the salt sections above the mining level. Solutioning of the salt could ultimately lead to weakening of the shaft lining, particularly if the solutioning was located differentially around the shaft lining. It is also important to know if water is present at any place where damage occurs to the shaft lining for various reasons - accidental, subsidence induced or natural (earthquakes).

It is recommended that a research program be initiated to develop non-destructive instrumentation and techniques to detect the presence of water behind the concrete portions of the shaft linings. (Cast iron or steel tubbing was installed at water bearing formations).

Several technologies should be investigated for suitability. Radar, electromagnetic and perhaps sonic or acoustic techniques require evaluation for additional research and development to resolve the problem of detecting water behind the concrete shaft liner.

4.2 RECOMMENDATIONS FOR RESEARCH AND DEVELOPMENT RE: MINE INDUCED SEISMICITY

The recent development of earth tremors related to potash mining has demonstrated the necessity of research to determine the cause of this phenomenon, the mechanisms by which they occur, and the probably maximum magnitude of the worst case event.

Gendzwill (1983) has proposed a theory for the "source mechanism" of the Cory mine earth tremors. He postulates that "the mechanism ..... appears to be a horizontal shear failure along planes of weakness in the carbonate rocks over the mine", and that "Shear failure is due to flexure and subsidence of the overburden". This theory is, in part, supported by the fact that no visible damage or movement has been measured or observed in the mining level. As the potash horizon is underlain by about 100 m of salt with minor anhydrite, the inescapable conclusion is that the

earth tremors are occurring in the rock formations above the mining horizon. This, in turn, narrows the number of factors to be considered in a research program to resolve the problem of mining induced seismicity.

4.2.1 MECHANISM OF GENERATING MINING INDUCED SEISMICITY

There is general agreement that the primary mechanism is a build-up of stresses in the overlying formations, which are then relieved by fracture or failure of the rock when their inherent strength is no longer capable of withstanding that stress. The most obvious agent of the mechanism is subsidence. Subsidence is influenced by mine design, mining pattern, extraction ratio, rate of extraction and composition of the overburden. All of these factors are interrelated. Consequently, any research into induced seismicity must include all of these factors.

4.2.2 MINE DESIGN

Mine design in the context of induced seismicity, is the layout and sequence of excavation of entry systems and production panels. It is dictated, to a great degree, by:

a) mine safety - protection of the miners and the integrity of the mine;

b) economics - cost of mining machines, conveyor belts and related equipment;

c) production requirements - tonnes of ore;

d) geology - ore grade, avoiding hazardous areas;

e) mine planning - short (1-5 years) and long range plans (5-10, 15+ years); and

f) advances in mining technology.

Mine design is also intimately related to the mining pattern and the mining system used to extract the ore. Another section of this report focuses on mine design, including the mining pattern and systems used.

In this section, the recommendation for research into Mine Design is specifically restricted to the cause, prevention and/or elimination of induced seismicity. Two possible contributing design factors should be investigated:

- the effect of the concentration of mining activity in one geographic area of the deposit, and

- advance and/or retreat mining.

a) Concentration of Mining Activity in One Area

Any number of circumstances (lack of planning, equipment, etc.) can force the concentration of mining operations to be carried out in one geographical area of the mine as happened at the Cory mine. (Three mining machines operating in three adjacent panels). This concentration of excavation in a confined area over a shorter period of time as compared to the older adjacent areas may cause a stress build-up in the overlying formations that results in an earth tremor. This factor should be researched to determine its effect.

b) Advance and/or Retreat Mining

The original mine design for all Saskatchewan's potash mines utilized the concept of advancing the mining from the shafts to the property boundaries. Several mines in the Saskatoon area have now changed to the retreat mining method. Development entries are now driven to the boundaries and the ore is then mined in panels in a retreating sequence back towards the previous mined out area closer to the shafts.

It is assumed that the "old" mined out area will or has stabilized to a uniform rate of closure. Will the retreat mining, as it approaches the mined out area, reload that area and result in induced seismicity? This question requires research.

4.2.3 MINING PATTERN

The chevron style of mining is suspected to build-up stresses in localized areas above the mining level. The room and pillar is thought to distribute these stresses more uniformly. This whole area requires research.

4.2.4 EXTRACTION RATIO

The present practice in Saskatchewan potash mining appears to revolve around a maximum extraction ratio of 35-40% for overall mine safety and integrity. This figure is thought to have originated from the experience of the New Mexico operations. Until the advent of the seismic events, this figure seemed appropriate. Is the 35-40% extraction ratio still appropriate?

This question becomes more significant when considering the following changes in mining designs being tested or developed:

a) Two Layer or Bi-level Mining

Several mines in the Saskatoon area are developing test areas for mining in the potash zone immediately below the present mining level (for example, see Schmitke, 1983). The present plan is to mine under previous mined out panels. This will increase the extraction ratio. The possible effects on the safety factor should be researched.

b) Total Extraction Mining

PCS Mining is at the present time investigating the feasibility of utilizing Total Extraction Mining at Rocanville (Taylor, 1983). Although the program is still in the feasibility study phase, it is expected that the extraction ratio would increase dramatically, with the use of back-filling. A considerable amount of research in all the facets of this novel mining technique has been carried out by PCS Mining, R & D Group, to include material handling, backfill implacement, geomechanical properties of salt backfill, rock mechanics, appraisal of the system and the instrumentation of the proposed panel. Recently, in conjunction with CRM and CCP, a short program of research to study the in-situ behaviour of salt backfill was initiated at CCP under CANMET contract No. OSQ83-00323. As this technique could have a great repercussion on the reserve and percent extraction of potash ore, it is proposed that concentrated effort in the area of R & D for full utilization of this or similar systems of mining be carried out with financial backing from Federal Agencies.

4.2.5 RATE OF EXTRACTION

The rate of extraction should have a direct effect on the rate of subsidence and the rate of build-up of stresses in the overlying formations. In fact, it is thought that some of the earlier earth tremors at the Cory mine were related to the fact that three mining machines were operating close together in one area. This would have the effect of increasing the rate of extraction in a "confined area".

On a broader scale, if there was a dramatic increase in the potash market, then there would be a corresponding increase in the rate of production to meet this demand. What effect, if any, would this have on induced seismicity? This question should be addressed.

4.2.6 SUBSIDENCE

As stated previously, subsidence is considered to be the key factor in the mechanism of generating induced seismicity. Subsidence, in turn, is influenced by the factors discussed above.

Noranda Research Centre (CRN) carried out a study of potash mine subsidence as part of this project. Their report is incorporated into this report as Volume II. The reader is referred to CRN's report for its conclusions and recommendations.

4.3 RECOMMENDATIONS FOR RESEARCH AND DEVELOPMENT RE: GEOLOGICAL ANOMALIES

There is a requirement for research and development of instrumentation and techniques to locate anomalous geological conditions ahead of the mining face, and to accurately predict their potential hazard, if any, to the mining operation.

Most of the geophysical techniques as developed for oil and gas exploration and for "hard rock" exploration and mining are applicable to many of the problems encountered in potash mining. Seismics and electromagnetics are being used at this time. Borehole geophysics in horizontal drill holes is being studied. Other types of geophysical techniques, such as radar and sonar, which have been used with some success in salt mining, are applicable to potash mining.

Unfortunately, all of the geophysical methods that have been tested or are being utilized in potash mining have moderate to severe limitations of range, discrimination, resolution, interpretation, reliability, ease of operation and/or cost. Further research and development of these techniques is recommended to reduce these limitations to acceptable levels.

It should be noted that many of the techniques applicable to the problem of "Geological Anomalies" are also applicable to the problem of "Water Inflows", including measurement of salt back thickness.

4.3.1 SURFACE SEISMIC SURVEYS

Surface reflection seismic surveys, as developed for oil and gas exploration, are now employed as a "routine tool" in the potash industry. These surveys have proven to be capable of outlining the large geological anomalies which are then avoided in mine planning, as well as outlining the broad, relatively flat areas of Prairie Evaporite which are considered to be indicative of good mining conditions. With the exception of

the large anomalies, seismic evaluation of the mining area is capable of outlining potential problem areas, but cannot as yet pinpoint the location, extent or type of problem to be expected. The reasons for this lack of adequate specificity are:

a) Discrimination and Resolution

The resolving power of seismic is dependent on the wave length and frequency of the seismic wave, as well as the density contrast and thickness of the layer to be "measured" or mapped. For example, for practical purposes, it is not possible to detect the potash layers in the Prairie Evaporite primarily due to a lack of density contrast between potash and salt. For all practical purposes, it is impossible to "map" the Dawson Bay Formation as its thickness (30-50 m) is insufficient to provide an adequate seismic response. The seismic reflection appears to be a composite of the underlying Second Red Bed, the Dawson Bay itself and probably the Hubbard Evaporite and First Red Bed (P. Bediz, personal communication).

The top of the Prairie Evaporite Formation can be mapped with a high degree of reliability. However, the accuracy of the depth to the top of the Formation, which should be on the order of $\pm$ 1 to 3 m, as at the Cory mine, is variable at other locations. This is, in part, due to the presence of surficial sand and gravel deposits, as at Rocanville, and to possible changes in the "character" (facies changes, thickness changes) of the formations above the Prairie Evaporite in different areas.

b) Cost

The average cost of surface seismic survey is now around $5 000 per line-mile in the potash mining area. The present size of surveys carried out by the individual mines is on the order of 20 line-miles at around $100 000. This size of survey provides coverage, at a one mile line spacing, of the approximate area to be developed and mined over the next five years. The one mile line spacing is capable of locating the larger geological anomalies but is insufficient to give the necessary control or line spacing to detect the smaller anomalies (less than 100 m in diameter). In order to pinpoint the location of the smaller anomalies, the line spacing would have to be reduced to 30-60 m. The cost is prohibitive for potash mining.

It should be noted that both dynamite and Vibroseis methods are used in potash seismics, with dynamite being the predominate method. Vibroseis, which is more expensive, is used at Rocanville for two reasons:

1) The high cost of drilling shot holes in the gravels closely approximates the increased cost of Vibroseis.

2) The ability of Vibroseis to generate a wave train of controlled frequencies has increased the reliability of the seismic results at Rocanville.

4.3.2 RECOMMENDATIONS FOR RESEARCH AND DEVELOPMENT OF THE SEISMIC SURVEY

The present methods utilized, dynamite in shot holes and Vibroseis, are very expensive when viewed in the context of potash mining. It would appear, however, that these are the only methods that can penetrate to the required depths, 1 000 - 1 500 m. It may, however, be worthwhile to have outside experts examine and evaluate the procedures utilized, both in data acquisition (field shooting) and data processing (computer) to determine if significant cost savings could be realized by changes to the normal oil and gas seismic techniques presently in use.

The second area recommended for research is the frequencies used in potash seismic surveys. The University of Saskatchewan, under the auspices of PCS Mining, has been carrying out limited research on the use of higher than normal frequencies to improve resolution. The results to date, although considered by some to be promising, are also considered to be controversial. More research is recommended in this area.

4.3.3 ELECTROMAGNETIC METHODS

This geophysical method is described under "Water Inflows" in this report. The research and development of a reliable EM system for "Water Inflows" would assist in the location of anomalous geological conditions that contain water, and would enhance the determination of the <u>possible</u> water hazards posed by "Geological Anomalies".

4.3.4 RADAR SURVEYS

The utilization of this method is recommended for detection of geological anomalies and water inflow. (This section is discussed in a later part of this report.) Although the instrumentation tested to date has limitations of range, resolution, discrimination and ease of operation, it is recommended that research and development be carried out to overcome these limitations.

4.3.5 SONIC AND SONAR SYSTEMS

As discussed under "Water Inflows", this technique has not been tried in Saskatchewan potash mining. It is recommended that this technique be researched and developed to help resolve the problem of identifying geological anomalies ahead of the mining face. It is probable that sonic or sonar would have to be combined with radar to produce the optimum instrument to detect geological anomalies.

4.3.6 BOREHOLE GEOPHYSICS

PCS Mining has initiated a study to determine the feasibility of using borehole geophysical systems - radar, electromagnetics, gamma ray spectrometry and seismics - in horizontal drill holes in the mining horizon to locate water and other hazards (anomalous areas) ahead of the mining face.

The Cleveland potash mine at Boulby, England, has demonstrated that directional horizontal drilling to lengths of a kilometre is possible using a conventional diamond drilling rig. Boliden of Sweden have developed an "in hole" radar system. University of Toronto has developed an "in hole" seismic system. Both of these systems will be tested in the PCS Mining study as well as standard gamma-ray logging. Various "in hole" EM systems as developed for hard rock exploration are being investigated for possible use in potash.

It is recommended that this area be further researched and developed for application to potash mining. It is expected that the system can be developed for the detection of geological anomalies, water above and ahead of the mining face, and, hopefully, determination of salt back thickness.

4.4 WATER INFLOWS

There is a requirement for the research and development of instrumentation and techniques to determine the presence (or absence) of water and its areal extent both above and ahead of the mining operation. There is an additional requirement to research and develop instrumentation and techniques to determine, with a preferred accuracy of ± one metre, the salt back thickness above and ahead of the mining operation.

General techniques have been tried, with limited or questionable success, to solve the two problems of water and salt back thickness. Other techniques are being investigated to determine if they can be applied to solve these two problems. These techniques are described below, with recommendations for additional research and development.

4.4.1. Surface Seismic Surveys

In the case of water inflows, seismic surveys cannot detect the presence of water, nor can they determine thickness of the salt back. However, Gendzwill, by using geological inferences from seismic records, i.e. the lack or presence of Davidson Evaporite, has inferred the presence or absence of brine in the Dawson Bay (Gendzwill, 1985). These surveys are used to determine if there are any unusual changes in the top of the Prairie Evaporite which may indicate an area where salt back thinning may occur or where water inflows could occur. For a detailed discussion of the limitations of the seismic technique, and recommendations for research and development, please refer to the section on Geological Anomalies.

4.4.2. Electromagnetic Surveys

Electromagnetic surveys have been used with some limited success in potash mines to map salt back thickness, and to detect the presence of water, above the mine workings (Gendzwill, 1983). The result of actual surveys indicate that for a thick salt back (over 15 m, as at Rocanville) the accuracy is within two or three metres, using an EM 34-3 instrument and a two layer model. The same instrument has been used with some apparent success to map water in the Second Red Bed above the salt.

Gendzwill (1983, p. 145) has outlined some of the problems encountered with the system - metal objects such as mining machines, power cables, conveyor belt systems in the survey area; water on the floor or in cracks in the roof and floor; shale content of the salt; intergranular water in the salt.

The accuracy of the system (± 2 to 3 m) is less than the desired ± 1 m. The development of a three layer model is expected to enhance the accuracy for thin salt back (less than 15 m).

The EM 34-3 system can only be used in existing excavations. It is not capable of "mining ahead" of the mining face.

It is recommended that research and development of Electromagnetic methods be carried out. The EM 34-3 is a system that could possibly be improved; but other systems may exist that are better. The whole field of EM geophysical systems, including from surface, should be examined to determine the suitability of methods to solve the potash mining problems.

4.4.3. Radar Surveys

Underground radar probing of coal, salt and potash has been carried out in the past with promising results (Unterberger, 1979; Fowler, 1983). The Geological Survey of Canada, Resource Geophysics and Geochemistry Division carried out a radar test in Saskatchewan potash mines in 1977 (Annan and Davis, 1977).

Attempts were made to determine salt back thickness and presence of water, with questionable success.

The present technology has limitations of range, resolution and discrimination. Additional research and development is recommended.

4.4.4. Sonic and Sonar Systems

These systems have not been tried in a potash mine. Unterberger (1979) reported on a system used in salt mines. It apparently is useful in both wet and dry salt, whereas radar has difficulty in salt that contains moisture.

It is recommended that research and development be carried out using sonic and sonar systems in potash to resolve the problem of salt back thickness and detection of water above, below and ahead of the mining operation. It is further suggested that this program include a study to determine if radar combined with sonic or sonar will produce a useful tool for potash mining.

4.4.5. Underground Seismic Surveys

Underground seismic surveys are reported to have been carried out with limited success. The fact that these surveys require shot holes into the salt back, and the use of small dynamite charges rule out their further development.

4.4.6. Oxygen Isotope and Chemical Analyses Studies

The University of Saskatchewan in Saskatoon has recently established facilities for analysis of stable isotope ratios under the direction of T. Kurtis Kyser, Assistant Professor, Department of Geological Sciences. PCS Mining has utilized this new facility to analyze some of the oxygen isotope ratios of the water inflows they have experienced in their mines.

Samples of water entering the shafts have been analyzed and used as references for certain formation waters. There is always the thought that such samples may not represent that specific formation if the sample was taken from a seepage or leak in the shaft lining.

Despite this doubt that may exist, the results to date have demonstrated that the oxygen isotope ratio analyses can positively identify waters of different ages. It is recommended that a research program be set up to collect water samples from all potash mines in Saskatchewan for oxygen isotope ratio analyses. This would include samples from the different shafts as well as from the various water inflows experienced by the different mines. This sampling would include detection of changes in composition and age over time.

In addition to the sampling of the various mines, it is recommended that, where feasible, formation water samples be obtained from a few selected future exploration wells, including oil exploration wells, drilled in or near producing potash mines, to obtain reference oxygen isotope ratios. In particular, this may be feasible at Kalium Chemicals potash mine where they routinely drill production solution wells.

Every potash mine, as a routine practice, chemically analyze water inflows. Although such analyses have limited value when attempting to determine the source of the water, the analyses do determine if the water is "saturated". Unfortunately, the analyses do not indicate if the chemical components are from the original source or have been dissolved from the salt cover through which the water has travelled. A study of chemical analyses of the various water leaks, combined with oxygen isotope ratios, may resolve this question.

4.4.7. Borehole Geophysics

PCS Mining has initiated a study to investigate the feasibility of utilizing borehole geophysical systems in horizontal drill holes in the mining level to locate water and other hazards ahead of the mining face. Such a system may be adapted to determine salt back thickness and the presence of water above or ahead of the mining face. This system is discussed in detail under "Geological Anomalies" in this report.

4.5 BRINE INJECTION WELLS

The most obvious approach to determining what is happening to the injected brine in the disposal formation would be to drill several wells into the formation beneath the mine and take physical and chemical measurements. The cost of each well, estimated at a minimum of $250 000, is considered prohibitive at this time. The fact that such wells would have to penetrate through the mining horizon is in itself considered to

be introducing a certain risk to the operation. This is in part due to the probable close proximity to the mining area (i.e. - within the mining lease).

There are other approaches that warrant research prior to drilling exploratory wells.

4.5.1 STUDIES OF CASE HISTORIES

Although brine injection wells have been in use in the potash industry since 1968, there does not appear to have been any study carried out on the injection wells. Such a study is recommended and should consist of a case history of each disposal well.

Such a case history would comprise a summary and comparison of the following:

a) Construction

Particular attention should be paid to the configuration of the bottom of the well where the injection occurs. Is the perforation of the pipe slotted or shot-perforated as in production oil wells? What type of seals or packers are used to seal off the disposal formation? Is the well double cased from surface to the disposal formation? What precautions or detection systems, if any, have been installed to detect failure of the system?

b) Operation and Usage

A detailed survey of the quantities of a brine injected over time (daily, weekly, monthly, annually, cumulative) should be compiled for each of the potash plants. A similar survey of operating pressures, injection rates and results of annual pressure tests should be compiled. The results of these surveys should then be compared to the different configurations of the injection wells and disposal depths.

c) Maintenance

A survey of the problems encountered in maintaining the various brine injection wells should be carried out to pin-point potential failures in the system. This would include problems associated with plugging of the disposal formation, techniques to increase injection flow rates, or changes in procedures that have been introduced since the wells were completed.

An analyses of the results of the study of these case histories would detail the problems encountered in the use of brine injection wells by the potash industry.

4.5.2 COMPARISON TO OTHER INDUSTRIES

Several industries use injection wells to dispose of liquid wastes. The oil industry and the chemical industry are the two largest users of

injection wells, and have been for a longer period of time than the potash industry. It would be logical to assume that these two industries should be a source of historical data that would be of value to determining the possible threat to potash mine stability from brine injection wells.

The results of the study of case histories of brine injection wells in the potash industry and the problems encountered should then be compared with the experience of liquid waste disposal wells utilized in the chemical and oil industries, assuming such information is available in this area. This comparison may lead to solutions to problems with brine injection wells or to areas of potential beneficial research.

4.5.3 DRILLING TEST WELLS

It is probable that the studies recommended above will ultimately lead to a requirement to drill test wells into the disposal formations to determine what effects brine disposal is having on the formation. The location of such test wells will have to be co-ordinated with the mine staff at the properties to be tested. The report of Simpson and Dennison (1975) on "Subsurface Waste Disposal Potential of Saskatchewan" should be utilized where appropriate to locate such test wells.

4.6 POTASH EXPLORATION WELLS

a) Reliability of the cement plugs

The only program suggested to test the reliability of the cement plugs in abandoned exploration wells is to drill out a number of selected old wells to arrive at a statistical figure for the integrity of the cement seals.

b) Location of exploration wells underground

Tarantolo and Unterberger (1978) have successfully located cased boreholes in salt. This suggests that radar be researched and developed to accurately locate abandoned exploration wells ahead of or adjacent to the mine workings. The fact that these holes will be filled with cement (or void) rather than steel casing will require additional research.

5. MINING METHODS AND ROCK MECHANICS CONSIDERATIONS

5.1 MINING METHODS

5.1.1 GENERAL DESCRIPTION OF MINING METHODS AND THEIR DEVELOPMENT

Since the inception of the Saskatchewan potash industry in the early 1960's, in consort with the equipment manufacturers, continual improvement in the advancement of the potash technology has been sought. Significant improvement in areas of operation have consequently been achieved, exemplified in the development of the multi-rotor boring machine, extensible and bridge-type conveyor systems, etc. Some breakthroughs in areas of rock mechanics and strata control have been attained, which has culminated in the present mining geometry of pillar and panel and chevron-type mining development systems. In spite of these advances, the industry cannot claim to have been able to date to safely increase the overall extraction of the potash ore beyond the 45% mark. It is pertinent to comment that the potash industry has been generally very slow in developing innovative and successful mining systems to achieve the above objective.

Table III shows the mining methods used by Saskatchewan potash producers. Initially, all the underground potash mines, with the exception of Rocanville Division, began mining with the room and pillar method, being the technology transfer from potash mines in the Carlsbad

area of New Mexico and from West Germany. The application of this system was unsuccessful in safe and productive mining of the potash, and with the exception of the Potash Company of America which continued with this system, the others modified the mining method to suit the different geological environment under which they were to produce. Some of the reasons for the unsuccessful application of this system could be explained as follows:

a) The deeper mining level, resulting in higher ground pressure.

b) Non-homogenity of the seam due to the presence of numerous clay bands which, after mining, as the result of interplanar reduction in cohesion, culminated in ground falls.

The application of this mining system caused unstable ground conditions, which could not be mined safely and economically.

TABLE III

Mining Systems Applied in Saskatchewan Potash Mines

	Initially	Presently
IMC	Room & Pillar	Long Room & Pillar
Rocanville	Long Room & Pillar	Long Room & Pillar
Lanigan	Room & Pillar	Long Room & Pillar
Central Canada Potash	Room & Pillar	Stress Relief
Allan	Room & Pillar	Stress Relief
Potash Company of America	Room & Pillar	Room & Pillar
Cory	Room & Pillar	Stress Relief
Cominco	Room & Pillar	Stress Relief & Long Room & Pillar

The unsuccessful application of the room and pillar mining method led to the development of the chevron and long room and pillar mining. This initiated some research activities in development of other mining methods which will be discussed under a separate heading.

The chevron pattern, commonly referred to as stress relief technique, was initially experimented with at Duval Corporation (now PCS Mining, Cory Division) in 1969. Testing consisted of extensive rock mechanics instrumentation of individual panels and eventual simulation of the panel using Finite Element Analysis and validation of the model to actual in-situ measurements. This evaluation indicated that there would be:

- the creation of stable access entries at the expense of the wing entries;
- occurrence of large magnitude of closure of the opening;
- concentration of a high extraction in the panels;
- the possibility of over-flexure of the overlying brittle carbonate formation in the form of micro-seismic acoustic emissions.

The full scale application of the chevron or stress relief method after this evaluation was adopted by some producers which proved the safe and more economical nature of this method of mining. The application of this method thereafter was adopted by other producers (Table III).

In spite of the developments of the above methods, the percent extraction of recoverable ore is still low and the mining costs are not matching the rate of inflation. The industry must adopt a rigorous Research and Development program to examine alternate mining methods, which would be safer and more economical overall than the present methods. This Research and Development program must specifically define the needs, objective priorities and the method of execution, which, in the majority of cases, would be site specific.

The potential methods of mining which render themselves for consideration are:

a) Bi-level mining,

b) Optimization of mining extraction (OME),

c) Backfilling in conjunction with present or future mining methods,

d) Longwall and shortwall mining.

Only the most amenable methods for Saskatchewan potash from "a" to "c" will be discussed in this context.

It is prudent at this stage to discuss some of the work currently undertaken in these areas by some of the producers, and establish the required area of further research.

5.1.2 BI-LEVEL MINING

In the Saskatoon area the potash occurs in two distinctive zones viz "A" zone (Saskatoon) and "B" zone (Lanigan Division). All the mines, with the exception of Lanigan Division which mines "B" zone, mine "A" zones. However, in the last few years, a positive move has been made in mining the "B" zone at Allan Division and Cory Division. With Lanigan Division giving consideration to mining the "A" zone overlying the main "B" zone workings, Cory Division excavated some 3 km of roadway in "B" zone. Allan's "B" zone excavation is around 20 km, while Lanigan Division has excavated a 20 ft high room in the "A" zone. To date, the experimental mining of the "B" zone in the case of Allan and Cory Divisions and monitoring of the ground condition has indicated a lower rate of convergence of the "B" zone and attainment of an early state of the stability in this horizon. The experimentation has indicated that this long term stability is achieved when "B" zone workings are mined below the old "A" zone workings i.e. the "A" zone workings have reached a state of stability, and hence allow the lower "B" horizon to be excavated in relaxed ground. It must be emphasized that the long term stability of "B" zone is only ensured when the "B" zones workings underlie the existing older "A" zones.

The future area of investigations with the bi-level mining method would be the optimum geometry of the "B" zone mine workings. From the "B" zone drivages at Allan Division and monitoring of the deformation, it has been categorically established that it is imperative to superimpose the two zones above each other to avoid any detrimental long term

interaction between the mining horizons. Currently, work is being undertaken by PCS Mining Allan and Cory Divisions in determining the best geometry of "B" zone workings. Only by concentrating the efforts in a comprehensive program of rock mechanics instrumentation which is divorced from the daily operation, the most expeditious and comprehensible data become available for analysis and consequent implementation of this mining method in future development.

5.1.3 OPTIMIZATION OF MINING EXTRACTION (OME)-TOTAL EXTRACTION MINING (TEM)

If the innovations in the field of rock cutting and material handling which were developed by the potash industry could be utilized in conjunction with a mining method then the desired level of extraction ratio could be achieved. This leads to the idea of OME (originally TEM) method. This refers to the continued use of four-rotor mining machines and extensible conveyors and maintaining a room length of up to 1 800 m, and 6 m wide, which in comparison to a mechanized longwall face would yield some 60-80 times the quantity of mined ore. (Taylor et al, 1983)

The initial study on the geochemical behavior of the ore and surrounding strata suggested that such a system could be moved sideways to constitute an OME (TEM) system. The success of the system depends on the successful stowing of salt tailings in the waste area at a sufficient in-place density to limit and strictly control closure of the mined-out opening. The advantages and disadvantages of this system are listed by Taylor (1983) and are briefly outlined here:

Advantages:

- reduction in environmental hazard due to the accumulation of salt tailings on surface,

- cost savings as a result of reduction in the volume of dumped salt tailings,

- increase in extraction ratios and longer reserve life,

- reduction in capital expenditure in material handling equipment due to the concentration of the output from a small area.

Disadvantages:

- preparation and transportation of the tailings would constitute an additional cost in the mining operation and higher capital cost,

- more intensive management effort.

The application of this concept to potash mining requires extensive Research and Development effort to explore the parameters of the problems and to define these for evaluation. This concept has been undergoing an extensive study at PCS Mining, and the problems addressed by Research and Development programs could be subdivided into the following:

a) Risk analysis study,

b) Vertical transportation of the salt tailings,

c) Earth Sciences program, including numerical analysis of the proposed panel, in-situ instrumentation capable of transmitting data over 3 000 m, geophysical monitoring using mining induced seismicity and the study of the geomechanical behavior of the salt tailings under varied stress conditions,

d) Stowing of tailings,

e) Conveyor development.

Work has been done in all of the above areas by PCS Mining and the concept is under review for a full scale demonstration. The utilization of this technique by other potash producers would necessitate site specific studies in the majority of the above areas and specifically in:

a) Numerical analysis study on the stability of the panel based on the geology of the site.

b) Research into the vertical transportation of salt tailings related to the mechanical and chemical characteristics of the tailings.

c) The required method of stowing - mechanical vs pneumatic.

What is evident is that the technology evolved from the concept of OME (TEM) could be used on a modular basis for improving the extraction ratio of the mined potash. For example, the salt tailings study and stowing of the salt underground could be treated as a separate entity and the mining methods, either novel or presently adopted, could be combined with this operation to improve the percent extraction and increase the overall stability of the opening.

The electrostatic separation technology as used by the European producers in retrieving the KCl from the halite (if carried out underground), and the rejection of impurities for stowing underground not only would improve the stability of the mine but would result in cost savings in the areas of:

a) Increased hoist capacity,

b) Substantial cost reduction due to reduction in the volume of tailings piles,

c) Less consumption of water and energy of drying in the refining process,

For example, hoisting capacity will be doubled and waste material left underground. Approximately 70 percent of present waste material is hoisted.

The longwall mining method has been used in Germany, France, Spain, and Russia in mining of potash. In the United States, attempts to apply mechanical longwall methods to the mining of Trona have been unsuccessful. The use of waste backfilling methods in conjunction with longwall operations has been employed by a few coal mines. This technique has been superseded by the advent of heavy powered support which provides positive support at the working face. Waste backfilling in potash mining has been used in West Germany and recently in a newly operational Potash Company of America mine in New Brunswick. No longwall mining of potash in conjunction with systematic salt backfilling is undertaken by any operator. If this system of mining is to be used in mining the potash deposits of Saskatchewan, a great deal of research in the area of support behind the advancing powered support should be undertaken. Obviously, the most logical candidate would be the salt tailings with obvious cost advantages as discussed earlier.

The technique involved in such a system is not proven and requires a considerable amount of research and development. It should be emphasized that the potash industry has achieved a great deal in a short span of time in the development of mining technology. However, a need exists in the potash industry for a dedicated research and development program by which to retain superiority in low cost production technology, and this could only be achieved by cooperation between the industry and the government.

5.2 ROCK MECHANICS CONSIDERATION

The science of rock mechanics and strata control has made its contribution to the successful mining of the Saskatchewan potash deposits at the depth of greater than 1.0 km. As discussed earlier, the deployment of other mining techniques to increase the percent extraction of this resource is highly desirable. What is fairly evident is that there are certain "gray" areas in the understanding of the geomechanical behaviour of the rocks surrounding the ore which require further investigation. This section will address certain specific areas but only those major areas of concern will be discussed, under the following main topics:

a) Mine opening

b) Intervening strata

c) Surface

5.2.1 MINE OPENING

The majority of rock mechanics investigations carried out in potash mining has been conducted in this area. The major part of this work was carried out by Serata since the early inception of the industry, which included in-situ instrumentation of the underground openings and the use of these data in formulation of a mathematical model using the Finite Element Analysis program-REM (Rheological Element Model). The proprietor claims the capability of the model to predict the deformation characteristics of the opening over a long period of time, in certain cases up to 10 000 years. What is evident is the lack of reference to the flexural

characteristics of the overlying competent strata. In previous chapters, the monitoring of what is believed to be the mining induced seismicity due to this phenomenon and possible hazards therefrom, was discussed. Although the role of mathematical models as a tool for design cannot be over-stressed, the future efforts in this area should address dynamic loading of these formations and base the future mine design on minimizing the effect of mining on the integrity of these formations.

A great deal of research in the area of seismicity has been conducted at the University of Saskatchewan (Gendzwill, 1983). Recently, PCS Mining has installed underground arrays in the Cory Division mine capable of detecting and locating the source of the seismic event and consideration is being given to extending this type of monitoring to at least one other mine in the near future. There is no need to stress the usefulness of this type of instrumentation in mine design, only suffice to mention that in the last two years, its importance has been magnified and attestation of its importance has been recognized by the industry and the research institutes. Two recent symposia on microseismic monitoring in relation to rockburst, and the formation of the CIM Rock Mechanics and Strata Control Sub-committee on "Microseismic and Rockburst Monitoring" is an indication of this level of awareness in the Canadian mining industry.

The rock mechanics study should be extended to include the determination of the geomechanical properties of the potash and other associated deposits by conducting a comprehensive laboratory study and also by further substantiation of this work by use of in-situ equipment, e.g. dilatometers, borehole strain gauges, etc. The large diameter triaxial

cells owned by the Saskatchewan Research Council should be utilized for this study. By the utilization of these data and their incorporation in a mathematical model, better use of this powerful tool for mine design could be achieved.

5.2.2 INTERVENING STRATA

An area which requires investigation is the determination of geomechanical properties of the overlying strata. Some work has been conducted by some researchers. Gendzwill (1983), Pandit (1983) and other unpublished work has been carried out by few of the companies like PCS Mining, but it must be emphasized that the data obtained are fragmented. On the other hand, the tests have been conducted on small diameter samples (1-1/2"), as well as the samples being prepared from an old borehole core, which had undergone the process of dehydrating and possible chemical change.

Due to the variable nature of the overlying Dawson Bay formation at different properties believed to be dry in the Saskatoon area, and a water-bearing formation in the southeastern part of Saskatchewan, a positive step must be taken in a comprehensive testing program, based on fresh cores to evaluate these fundamental parameters. Obviously, the varied nature of the strata suggests conducting tests on cores from each property which could be acquired by the coring from underground under strict controlled conditions.

Under a separate contract with CANMET (No. OSQ83-00323), CCP, PCS Mining and CRN, in a jointly sponsored projected with CANMET, are installing a number of stress meters and strain gauges in the overlying Dawson Bay, as a part of the grouting operation currently undertaken by CCP to seal the brine from their Panels 109 and 209. This investigation, on a similar line of approach, should be repeated at other properties, and the scope of investigation should include the mechanism of the deformation of the overlying formation in laboratory and in-situ conditions.

5.2.3 SURFACE

A major part of this report deals with this particular phenomenon of surface subsidence. The measurement of horizontal strain has been omitted from the subsidence surveys due to excessive costs, but should be incorporated in future surveys. This measurement, in conjunction with the investigations recommended in 5.2.1 and 5.2.2, would throw light upon the overall mechanism of the strata deformation due to potash mining.

6. GENERAL CONCLUSIONS

The general consensus of the various participants in the project is that the project has been successful in achieving its objective within the limits imposed by the quantity and quality of the geotechnical data available. The project has identified specific areas where research and development are required to ensure long term regional mine stability in the Saskatchewan potash mining industry. Areas where insufficient data are available to determine if a problem does exist or may develop in the future have been identified or, at minimum, indicated to exist, and thus require further research and probable development of instruments and/or techniques.

6.1 GEOLOGICAL RESEARCH AND DEVELOPMENT

During the course of the project, it became apparent that, although regional geology of the potash deposits is adequate, there is a significant lack of detailed geological information, particularly recorded data and maps of the individual deposits, and, of more significance, the geological anomalies encountered in previous mining. As most of the areas of occurrence may no longer be accessible, this information of past occurrences may be considered as "unavailable". It would be beneficial

to each mine to conduct a search of their files and assemble for permanent storage, any and all information available on past geological and mining "history". For the future, it is strongly recommended that detailed geological data be collected (maps, reports and photographs) and maintained on all geological or mining anomalies. This information would allow for detailed geological studies and comparisons of anomalies which, in turn, would then allow for comprehensive classification of the different anomalies, their probable origins, and, ultimately, their probable significance as a potential or real hazard, and methods to predict and avoid their occurrence.

6.2 GEOPHYSICAL RESEARCH AND DEVELOPMENT

Since the inception of potash mining in Saskatchewan, there have been tremendous advances made in the field of geophysical technology. It is the very strong position of the participants in the project that geophysical technology should be researched and developed for direct application to resolve specific potash mining problems as set out in this report. The following systems have been recommended for additional research and development:

a) Surface Seismic Surveys - to improve discrimination and resolution, and to improve cost effectiveness.

b) Radar Systems - to improve range, resolution, discrimination and ease of operation.

c) EM Systems - to improve range, resolution, discrimination and ease of operation.

d) Sonic, Acoustic, and Ultrasonic - to determine their applicability to resolving the problems indicated in this report, and to research and develop their potential.

e) Borehole Geophysics - to research and develop its promising potential to detect geological anomalies and water hazards ahead of the mining face.

6.3 GEOTECHNICAL RESEARCH AND DEVELOPMENT

The part of this project devoted to surface subsidence has shown that subsidence can be predicted to a reasonable degree with the data available. It has shown that more research and development is required, particularly in the areas of rock mechanics and the possible favourable effects of backfilling. It is the consensus of the participants in this project that such further research and development, particularly in the areas of backfilling and investigations of the mutually related effects of mine closure and subsidence of the entire overburden is both essential and desirable from the point of view of long term mine stability.

6.4 OPERATIONAL MINING RESEARCH AND DEVELOPMENT

Research and development of new mining designs or systems is a logical part of any program directed at improving mine stability. Such a program would involve examination of optimal extraction ratios and, of necessity, include geotechnical research and development as well as emphasis on increased productivity. Such a research and development program is strongly recommended.

In conclusion, this report has shown that there is a real requirement for research and development required to improve the regional mine stability in the Saskatchewan potash mining industry.

ADDENDUM

PCS MINING ROCANVILLE DIVISION

BRINE INGRESS

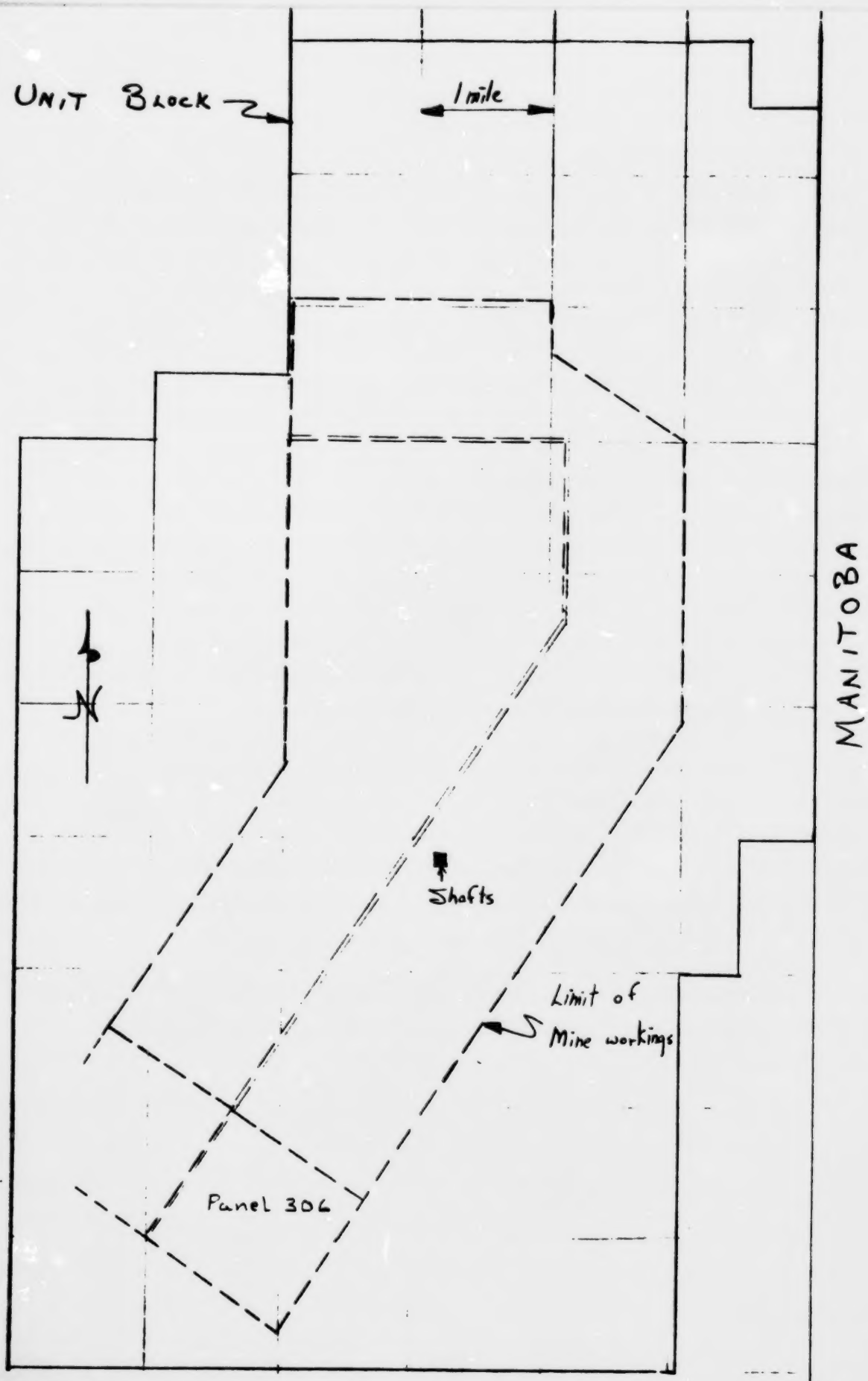
On November 18, 1984, PCS Mining Rocanville Division experienced a major brine ingress. Initial flow rates were estimated at about 3 000 L/min (660 gallons per minute). The highest subsequent surge in flow rate was estimated at about 19 000 L/min (4 180 gallons per minute).

At Rocanville Division, mining is carried out within the Esterhazy Member of the Prairie Formation (see attached stratigraphic section). While probing a salt horse, the four rotor Marietta miner, cutting 2.44 m (8 feet) high and 7.92 m (26 feet) wide, intersected a "collapse structure" with fragments of limestone. The mining machine was removed from the entry (see attached mine plans for locations). Six days later a significant brine inflow began.

Isotopic studies indicated a Dawson Bay-Souris River source for the inflowing brine. Periodic isotopic analyses have been done since the inflow began. These analyses have been very consistent. Each sample points to the Devonian carbonates as the source of brine.

Control measures included the preparation of an underground brine storage area and the installation of an up-shaft pumping system. A large bulkhead has been constructed to seal off the inflow. Flow rate through two 20 mm (8 in) diameter pipelines within the bulkhead was controlled at about 4 500 L/min (990 gallons per minute) at less than 1 380 kPa (200 psi) pressure.

Grouting operations are underway through one of the 8" lines. Further remedial action is dependent upon the success of the present grouting procedure.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Shafts
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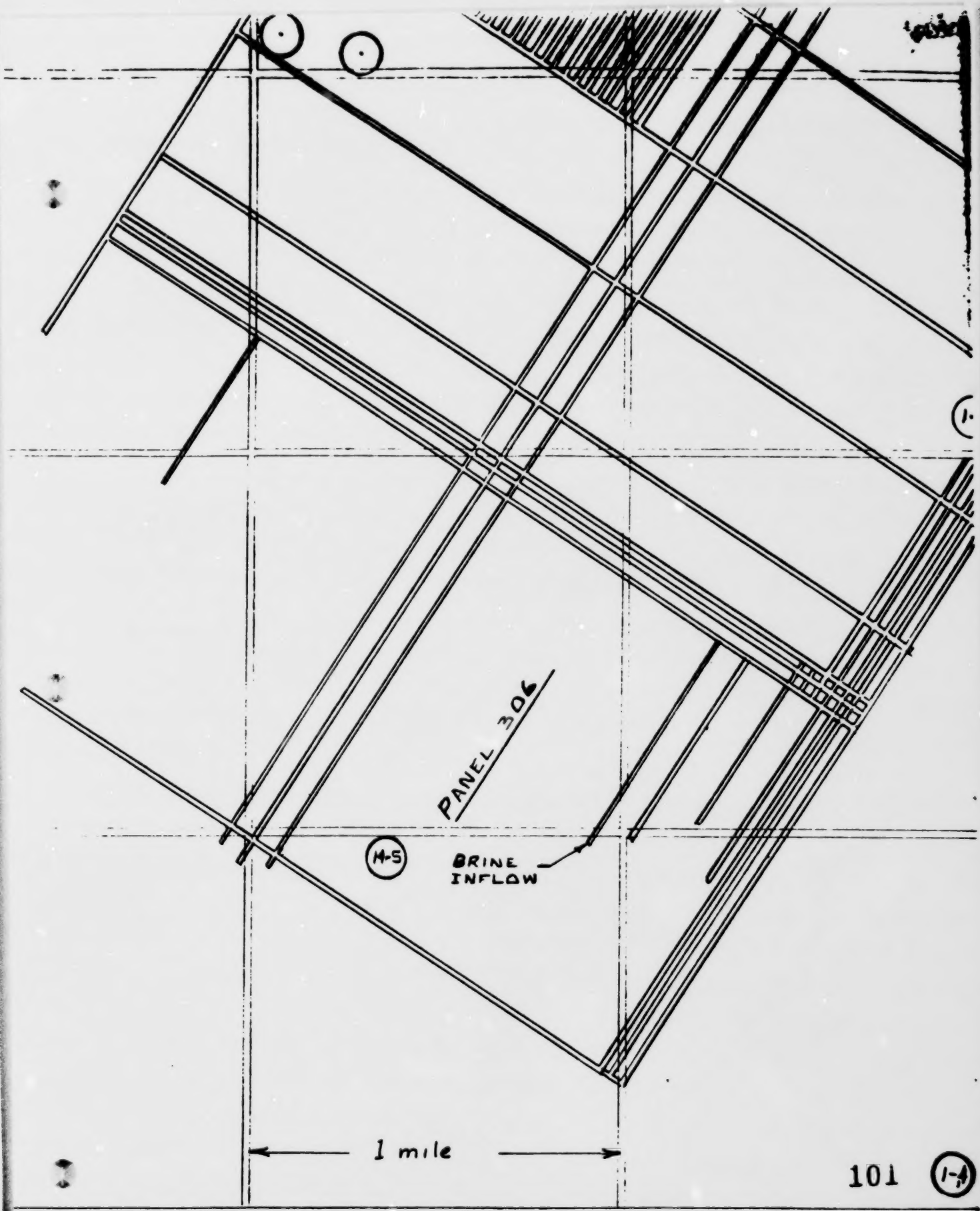
Mine workings
Panel 306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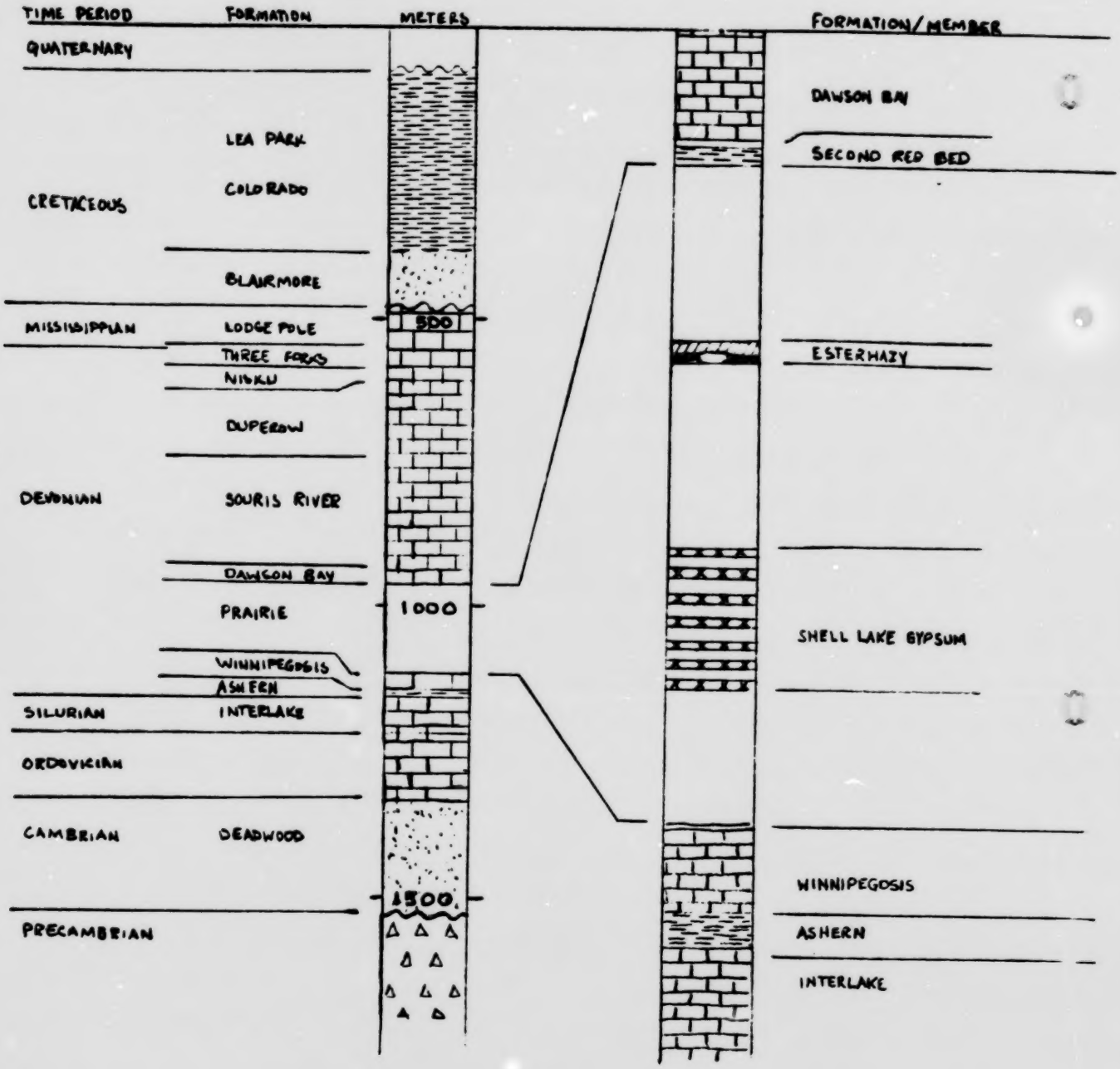
H-5
BRINE INFLOW
1 mile

STRATIGRAPHY OF PCS MINING ROCANVILLE



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APPENDIX "A"

QUESTIONNAIRES

REGIONAL MINE STABILITY PROJECT

SUBSIDENCE QUESTIONNAIRE

SUBSIDENCE QUESTIONNAIRE

REGIONAL MINE STABILITY PROJECT

S1. At what elevation below surface is the mining horizon?

Highest _____

Lowest _____

Average _____

S2. On what date did your mine commence underground operations?

S3. Please list the dates of EACH survey taken, including the initial survey, at your site, to include the time period during which the survey was performed. Please also indicate the frequency of measurements taken, and any comment on the validity or acceptability of each survey.

S4. To what order of precision was each survey conducted (First order, Second order, etc.)?

S5. For EACH of the surveys conducted, would you please provide the following:

a) The location of the datum in relation to the survey.

b) Map showing location of survey lines and stations.

c) Mine plans showing the time rate of mining advance relative to the subsidence surveys.

d) All available subsidence survey records of vertical deformation (i.e. movement or displacement), horizontal deformation, or both, for each subsidence station and/or monument.

e) Constructed subsidence profiles relating to mining rate, sequencing or time.

f) Equipment used.

g) Calculated strain and/or tilt data.

h) Mining panel and pillar dimensions and extraction ratio for areas relevant to each subsidence profile.

Note: Where dimensions of panels and/or pillars changed between surveys, please indicate the time and dimensions of the changes.

Please see question 19 re: extraction ratio.

S6. What is the maximum amount of subsidence that has occurred?

S7. In you opinion, does the subsidence record correlate with the mining sequence and/or extraction ratio?

Yes _____ No _____

Note: See also question S19.

S8. If answer to S7 is affirmative, is corroborating data available?

Yes _____ No _____

If affirmative, please supply same.

S9. What relationship is there between underground closure of the mine openings and surface subsidence?

S10. Can you provide corroborating data between surface subsidence and underground closure?

Yes _____ No _____

If affirmative, please supply.

S11. Please describe any visual effects of surface subsidence. (e.g. structural damage, drainage interference, etc...).

S12. Please describe any effects of surface subsidence on your surface and/or underground operations (e.g. tailings pile, pond, dikes, pipelines, etc...).

S13. a) In your mined out (abandoned) areas, do you have closure stations still accessible for measurement?

Yes _____ No _____

b) If answer is affirmative, please provide data on closure (magnitude and rate) relative to each survey carried out.

S14. Please provide the following information/data:

a) relevant geological (stratigraphic) sections and structural data for each subsidence profile.

If this information is NOT available, please provide the following:

i) geologic logs for shaft pilot holes, and/or

ii) geologic logs and sections from shaft sinking.

iii) contour maps and seismic sections from seismic surveys covering the mining area, to illustrate changes in thickness of overlying formations.

S15. Please provide any material on measurements of physical parameters for the overburden and the evaporite formations.

S16. Please provide the design parameters used for determining the shaft pillar size.

S17. To what depths are the survey monuments installed.

S18. Please provide diagrams of monuments used in each survey.

S19. Please define the term "Extraction Ratio" as used when answering questions S5.h) and S7.

REGIONAL MINE STABILITY PROJECT

EARTHQUAKE QUESTIONNAIRE

Part One - Earthquakes

Part Two - Microseismicity

Part Three - Brine Injection

REGIONAL MINE STABILITY PROJECT

EARTHQUAKE QUESTIONNAIRE

PART ONE - EARTHQUAKES

E1. Has your mining operation experienced one or more earthquakes?

Yes _____ No _____

E2. For EACH earthquake experienced at your operations, would you please complete one of the forms on the following pages E1-1 to E1-10.

E3. Has any correlation been made or inferred between the location and/or magnitude of the earthquake(s) and the following: (Please answer "made", "inferred" or "no")

a) mine opernings and/or mining pattern _____

b) numbers and/or locations of miners _____

c) panel sizes _____

d) room sizes _____

e) pillar sizes _____

f) brine injection _____

g) extraction ratio _____

h) other (please state) _____

_____

_____

EARTHQUAKE SUMMARY

Date and Time of quake _____

Magnitude: Unknown _____

Richter Scale _____

Mercalli Scale _____

Geographical epicentre in relation to shaft or mine workings:

Geological epicentre (Formation name or depth from surface):

Was the quake noticed or felt:

underground _____

on surface at the plant site _____

on surface off the plant site _____

NOT noticed or felt _____

Was the quake recorded on a seismometer or other instrument installed:

underground _____

on surface in the lease area _____

on surface off the lease area _____

DO NOT KNOW _____

E4. Do you have reports from your staff on the earthquake occurrence(s)?

Yes _____ No _____

E5. Do you have reports from consultants, universities, etc... on the earthquake occurrence(s)?

Yes _____ No _____

E6. Do you have records of the following, to be used to support any attempts to correlate earthquakes to mining methods? Please indicate hourly, daily, weekly, monthly, yearly or not available, as applicable.

a) Location of miners _____

b) production rate by miner _____

c) changes in pillar sizes _____

d) changes in panel sizes _____

e) changes in room sizes _____

f) changes in wing sizes _____

E7. Please briefly outline what steps, if any, you have taken to avoid creating earthquakes:

E8. a) Have you initiated any studies concerning earthquakes?

Yes _____ No _____

b) If Yes, please briefly describe the studies, and the organizations doing the studies:

PART TWO - MICROSEISMICITY

E9. a) Has any program been instituted or carried out at your operation to detect and/or measure microseismic events?

Yes _____ No _____

b) If affirmative, please provide a brief description of the type of study or studies, and whether "in-house" or by external organization:

E10. a) Can you provide any data on microseismic measurements at your operations?

Yes _____ No _____

b) If affirmative, please list report titles or type of data available:

PART THREE - BRINE INJECTION

E11. How many brine injection wells do you have?_____

E12. What percent of the time are they being utilized?

No. 1 _____

No. 2 _____

No. 3 _____

E13. a) Is there a specific season or time of year when they are used the least?

Yes _____ No _____

b) If yes, what season or time of year are they used the least?

No. 1 _____

No. 2 _____

No. 3 _____

E14. How far distant from the shafts are the wells located?

No. 1 _____

No. 2 _____

No. 3 _____

E15. How far distant, in the horizontal plane, are the wells located from present underground workings?

No. 1 _____

No. 2 _____

No. 3 _____

E16. Please provide a sketch or map showing the locations of the wells with respect to the shafts. Please indicate which well is No. 1, No. 2 and No. 3.

E17. Into what formation(s) is the brine injected?

No. 1 _____

No. 2 _____

No. 3 _____

E18. At what depth(s) is the brine injected?

No. 1 _____

No. 2 _____

No. 3 _____

E19. Can you provide data on the results of the annual pressure tests required by the provincial government?

Yes _____ No _____

E20. Can you provide data on the quantities of brine injected on a:

daily basis _____

weekly basis _____

monthly basis _____

annual basis _____

E21. Have you stopped using or abandoned any brine injection well?

Yes _____ No _____

E22. a) Since commencement of your operations, have you added new injection wells?

Yes _____ No _____

b) If yes, please explain why:

E23. a) Have you have to reperforate any of the brine injection wells?

Yes _____ No _____

b) If yes, please explain why:

E24. If you have repereforated, has it been into the same formation(s) used previously?

Yes _____ No _____

E25. If you have reperforated, was the reperforating done at a different elevation than previous?

Higher _____

Lower _____

Same _____

In-between previous perforations _____

E26. Do you have reports available describing the drilling and installation of the injection wells, including the "hardware" configuration within the well?

Yes ____ No _____

REGIONAL MINE STABILITY PROJECT

WATER INFLOWS QUESTIONNAIRE

Part One - General

Part Two - Exploration Drill Holes

Part Three - Shaft Stability

REGIONAL MINE STABILITY PROJECT

WATER INFLOWS QUESTIONNAIRE

PART ONE - GENERAL

W1. Do you have water/brine entering your mine workings at this time? (Shafts will be treated separately later).

Yes _____ No _____

W2. Do you have water inflow areas or locations that have now completely dried up? (Not even damp)

Yes _____ No _____

W3. Do you have water inflows in areas that are no longer accessible?

Yes _____ No _____

Don't know _____

W4. Did you have water inflows, presumably no longer active, in areas no longer accessible?

Yes _____ No _____

W5. Do you have chemical analyses of inflows that have occured at your site?

Yes _____ No _____

W6. a) Are the chemical analyses conducted on-site?

Yes _____ No _____

b) If off-site, please specify where they are sent for analyses:

W7. Please indicate what you analyze the water/brine inflow for:

Ca _____ Cl _____ K _____ Mg _____

Na _____ ph _____ Presence of gases _____

Other _____

_____

_____

W8. a) When an inflow has occured, do you routinely monitor it for change in rate of flow, change in chemical composition, presence of gases, etc...

Yes _____ No _____

W8. continued

b) If yes, at what frequency do you monitor:

W9. a) Do you maintain records on each of the water inflows:

Yes _____ No _____

b) If yes, Quantity only? _____

Chemical Analyses _____

Both _____

W10. a) Do you have reports available on each of the inflows?

Yes _____ No _____

b) If yes, are the reports by:

Mine Staff _____

Consultants _____

Both _____

W11. a) Have you determined the source of the water inflows?

Yes _____ No _____

b) If yes, please briefly state the basis of your determinations:

W12. a) Have you applied underground geophysical methods to any of the inflows?

Yes _____ No _____

b) If yes, please briefly describe the dates, method, purpose and results of each, and indicate if carried out by mine staff or others:

W12. b) continued

c) Are detailed or formal reports available for these methods/surveys/results?

Yes _____ No _____

W13. To what depths do you drill internal rock mechanics stations?

Into the back _____

_____

Into the floor _____

_____

W14. a) Do you have salt back thickness maps for your mining area?

Yes _____ No _____

b) If yes, how were they derived?

W15. a) Have you tried any system (drilling, geophysical, etc.) to determine from underground the salt back thickness in your mining area?

Yes _____ No _____

b) If yes, please briefly describe the system(s) used, purpose and results:

W16. Do you use surface seismic surveys as a "routine tool" in your operations?

Yes _____ No _____

W17. What pattern and spacing are the seismic lines?

W18. What formation tops do you map with your seismic surveys?

W19. What formations do you isopach map (i.e.-thickness) with your seismic suveys:

W20. a) As a routine procedure, do you attempt to correlate ore grade, mining conditions, occurrences of geological anomalies, water inflows, etc. to the seismic maps?

Yes _____ No _____

b) If yes, to which seismic maps do you correlate:

W21. How many years ahead of mining an area do you carry out seismic surveys?

W22. For the following, would you please indicate the SCALE of the maps maintained at your operation. Please indicate those maps that you do NOT maintain by N.E. for non-existence.

a) Seismic maps: Scale _____

Contour Inverval _____

b) Ore Grade Map _____

c) Carnallite Grade Map _____

d) Insols Grade Map _____

e) Underground Elevation Map _____

f) Location of Water Inflows _____

g) Location of Geological Anomalies or Disturbances _____

h) Location of Salt Horses _____ (undisturbed area where salt has replaced potash).

i) Please list, with scale, any other similar type maps you maintain:

W23. For your Underground Elevation Map, please specify:

a) What plane is mapped (e.g.-roof, floor, marker bed, clay layer, etc...)

b) What is the reference point of origin or datum?

W24. For those maps in question W22 that you have indicated are "non-existent (N.E.)", please specify those maps for which you have the data to allow their construction:

W25. With respect to the grade maps, would you please briefly describe the sampling technique used at your mine, including location of samples or readings taken, frequency and/or intervals between samples, etc...:

W26. a) Is the Dawson Bay (including the 2nd Red Bed) considered to be "wet" or "dry" over your mining lease?

Wet _____ Dry _____ Do not know _____

b) If wet or dry, how was this determined?

W29. Please indicate if the following formations are present or absent over your mining lease:

Formation	Present	Absent	Do not know
Hubbard Evaporite			
Davidson Evaporite			
Harris Evaporite			

W28. Do you have geological reports detailing the geology of your mining lease?

Yes _____ No _____

W29. Do you have geological repo_ts detailing the geology of specific parts of your mine, such as water inflows, geological disturbances, etc...?

Yes _____ No _____

REGIONAL MINE STABILITY PROJECT

WATER INFLOWS QUESTIONNAIRE

PART TWO - EXPLORATION DRILL HOLES

W30. Have you accurately located, on the surface, every exploration well in your mining lease?

Yes _____ No _____

W31. Have you surveyed in the surface location of every well in your lease area, and tied them in to your Mine Coordinate System?

Yes _____ No _____

W32. Can you provide a map showing the surveyed locations of the wells with respect to your underground workings?

Yes _____ No _____

W33. Have you determined the accurate location of the bottom of all wells drilled in your lease area?

Yes _____ No _____

W34. a) What is the diameter of the pillars left around wells drilled in your lease?

b) What criteria did you use to determine this diameter?

W35. Have you determined if the exploration wells drilled in your mining lease have been properly plugged when abandoned?

Yes _____ No _____

W36. Do you have the data necessary to determine if the abandoned wells were properly plugged?

Yes _____ No _____

W37. a) Have you ever mined into an abandoned exploration well?

Yes _____ No _____

b) If yes, was the hole cement- plugged?

Yes _____ No _____

c) If yes, was the cement tested to determine any deterioration?

Yes _____ No _____

W38. For each well drilled in your mining lease area, would you please fill in one sheet following. Please indicate if you have the listed data.

REGIONAL MINE STABILITY PROJECT

INVENTORY OF WELL FILES INFORMATION

WELL NAME AND LOCATION:	
ASSAYS OF CORE	ENGINEER'S REPORT
CHEMICAL ANALYSES OF DRILL STEM TESTS	SURVEYED LOCATION PLAN
CORE DESCRIPTION	SYNTHETIC SEISMOGRAM
DIRECTIONAL SURVEY	WELL COMPLETION REPORT
DRILL STEM TEST RESULTS	
LOCATION OF CORE:	

DOWN HOLE LOGGING	
ACOUSTIC OR SONIC LOG	ELECTRICAL LOG
BHC ACOUSTILOG	GAMMA RAY LOG
COMPENSATED DENSILOG	GAMMA RAY NEUTRON
COMPENSATED NEUTRON	INDUCTION LOG
DENSILOG	MICROLATEROLOG
DUAL INDUCTION FOCUSED	RESISTIVITY LOG

OTHER LOGS OR INFORMATION AVAILABLE

REGIONAL MINE STABILITY PROJECT

WATER INFLOWS QUESTIONNAIRE

PART THREE - SHAFT STABILITY

W39. Please indicate which shaft is Service, which Production:

Shaft No. 1 _____

Shaft No. 2 _____

W40. Please indicate any name given to the shafts (e.g. "Sarcee")

Shaft No. 1 _____

Shaft No. 2 _____

W41. What is the elevation of the shaft collars?

Shaft No. 1 _____

Shaft No. 2 _____

W42. What is the depth to the bottom of the concrete plug?

Shaft No. 1 _____

Shaft No. 2 _____

W43. What is the depth to the top of the concrete plug?

Shaft No. 1 _____

Shaft No. 2 _____

W44. How far apart are your shafts (centre to centre)?

W45. Please indicate which shaft is upcast and which downcast.

Shaft No. 1 _____

Shaft No. 2 _____

W46. Please indicate which shafts are considered to be "wet" and which are considered to be "dry".

Shaft No. 1 _____

Shaft No. 2 _____

W47. How much water is each shaft making at this point in time?

Shaft No. 1 _____

Shaft No. 2 _____

W48. a) Do you have data available to illustrate changes over time in the rate of water inflow to your shafts?

Yes _____ No _____

b) If yes, how long a period of time does the data represent?

W49. a) Do you have data available to illustrate the changes in chemical composition over time for the various leaks or inflows into your shafts?

Yes _____ No _____

b) If yes, how long a period of time does the data represent?

W50. Please briefly describe the method used to measure the amount of water entering the shafts.

W51. Please briefly describe how the water entering the shafts is collected and disposed of.

W52. a) Have you ever experienced any significant inflow of water into either of your shafts?

Yes _____ No _____

b) If yes, please briefly describe the incident:

c) Is a report of the incident available?

Yes _____ No _____

W53. At what frequency do you wash down the shafts?

W54. When washing down the shafts, do you use fresh water or brine?

W55. How often do you inspect the shafts

W56. Please briefly describe what precautions you take, when lowering oversized equipment down the shaft, to ensure the equipment does not damage the shaft lining:

W57. a) Has any object of significant size (e.g.-service cable, construction material, cage, skip, equipment, etc...) accidentally fallen down any of your shafts?

Yes _____ No _____

b) If yes, please briefly describe the damage, if any, to the shaft lining and/or concrete plug at the bottom:

W58. For each shaft, please list the type, quantity and size of service pipes or lines in each shaft. For the mine discharge to surface and the fresh water lines, please indicate if any are no longer used:

Shaft No. 1:

Shaft No. 2:

W59. What is the diameter of your shaft pillars?

W60. How was the diameter of the shaft pillar decided?

W61. Have you restricted the amount of mining or excavation in the shaft pillars?

Yes _____ No _____

W62. a) During shaft sinking, did you encounter any unexpected water inflows?

Yes _____ No _____

b) If yes, please briefly describe the problems:

W63. Do you still have complete records and reports of the shaft sinking operations?

Yes _____ No _____

W64. Are the pumping stations established during shaft sinking still accessible?

Yes _____ No _____

W65. At what depth and in what formations are the accessible pumping stations?

Shaft No. 1 _____

Shaft No. 2 _____

W66. a) Have you detected any solutioning around your shafts?

Yes _____ No _____

b) If yes, please briefly describe the effects detected. Include a sketch or photo if possible.

W66. b) continued ...

W67. Are there any reports available on the shaft solutioning including repairs or remedial actions taken?

Yes _____ No _____

W68. a) Since the time of acceptance of the shafts from the shaft sinking contractor, have you physically or chemically tested in any way, the deterioration, if any, of the steel and/or concrete linings?

Yes _____ No _____

b) If yes, briefly describe the method and results:

c) Is a report of the testing and results available?

Yes _____ No _____

W69. Since completion of your shafts, have you carried out any of the following rock mechanics measurements in your shafts?

a) Total Closure Yes _____ No _____

b) Creep Closure Rates Yes _____ No _____

c) Are the results of these tests available? Yes _____ No _____

W70. Do you have any reports of shaft inspections carried out by external consultants since construction of the shafts?

Yes _____ No _____

W71. a) Since the acceptance of the shafts from the shaft sinking contractor, have you carried out any major repairs to your shaft linings?

Yes _____ No _____

b) If yes, is a report available?

Yes _____ No _____

W72. Please briefly describe the shaft lining for each of your shafts, particularly indicating those portions lined with concrete, tubing, etc...

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VOLUME TWO

SUBSIDENCE PREDICTION FOR SASKATCHEWAN POTASH MINES

VOLUME TWO

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1. INTRODUCTION

In order to effectively control movement of the ground overlying a mining seam and prevent damage to overlying structures, an efficient method for the prediction of subsidence must be developed. As early as the beginning of the nineteenth century, subsidence theorys were postulated. An excellent review of the development of these theorys to present day is documented by Shadbolt(65).

The major subsidence prediction techniques at present may be divided into two major categories, namely Empirical and Analytical/Mechanistic techniques. Empirical techniques are based on experience gained through observation of subsided areas, while Analytical/Mechanistic techniques are based on the physical properties of the subsiding material and their reaction to changing mining geometries.

The empirical techniques are widely used in areas where years of observations have been recorded, and in some cases are claimed to be accurate to within 10% of observed subsidence (52). Analytical methods are limited by their complexity and the requirement that the subsiding materials physical properties be known, however with the increasing utilization of high speed computers these methods are finding more application.

The majority of subsidence prediction methods available have been developed for use with sedimentary, bedded deposits, particularly long wall coal mines in the U.K., U.S., and Europe, and are not readily applicable to room and pillar mining geometries and time dependent behavior as found in the Saskatchewan Potash mines.

2. TROUGH SUBSIDENCE DEVELOPMENT

Trough subsidence typically occurs with mining of thin, flat bedded deposits overlain by stratified rock. Examples of this type of formation commonly overly coal and potash mines (53). The general shape of the subsidence profile is shown in Figure 1.

Trough subsidence development above a longwall type mine typically occurs as follows; Mining of a panel initially results in downflexing of the roof of the excavation, with or without floor heave. As the excavation increases in area, roof sag increases until roof meets floor, or the roof fractures and falls. Surface subsidence, indicated by vertical and lateral movement, occurs at some time following the first sign of subsurface subsidence.

Methods used to limit surface subsidence include leaving unmined pillars, or artificially supporting an opening with props or fill. In some coal and non metallic mines, as the span of an opening is increased, fracturing of the back and roof falls occur, filling in the void volume. This fallen material eventually becomes compressed and partially or completely supports the overlying ground. In the case of room and pillar potash mines, with a thick overlying bed of salt , pillar support has time dependent behaviour and generally deforms at a slow rate. The overlying strata is allowed to deform by flexing,without fracturing. It is especially important to avoid fracturing of the back in Saskatchewan potash

mines as there is often a water bearing formation overlying the salt cover.

Many factors contribute to the subsidence process above thin bedded deposits. The interaction and stability of the floor, pillars and roof of a mine all effect the transferal of subsidence to the ground surface. A good summary of the factors contributing to subsidence above room and pillar mines is provided by Mathur and Mikkilineni (46). A summary of these contributing factors is presented in Figure 2.

The mine geometry and related factors such as seam thickness and depth, panel width, pillar width, extraction ratio, rate and direction of advance, face location, multiple panel mining, and presence of multiple seams, are considered to play an important and dominant part in the empirical methods of subsidence prediction.

2.1. SUBSIDENCE CHARACTERISTICS AND TERMINOLOGY

The general shape of the trough subsidence profile found over a flat seam excavation is shown in Figure 1. It's shape is generally symmetric about the centre of the excavation, providing the excavation geometry is simple and regular, and extends to some distance past the edge of the workings. Typical sections through workings illustrating standard symbols for subsidence, slope, strain and displacement are included in Appendix 1., along with subsidence terms and definitions from the N.C.B. handbook (52).

In some cases it was found that the subsidence profile that develops perpendicular to the mining direction is different from the profile developing parallel to the mining direction (49). If the subsidence has time dependency the profile of the subsidence curve overlying the travelling face will also be unique.

The subsidence development curve generally takes on the shape shown in Figure 3. When workings get within a critical horizontal distance from a point, subsidence at that point begins. Experience with U.K. longwall mines shows that when the working face is directly under the point only 8-15% of total subsidence will have occured. As the face passes by the point subsidence will accelerate rapidly . Time independent subsidence will approach a maximum when the face has travelled a critical distance from the point.

2.1.1. COMPONENTS OF THE SUBSIDENCE CURVE

2.1.1.1. Vertical Displacement

The vertical displacement of a point on the surface tends to be a maximum at the centre of a panel, reducing to zero at a distance past the edge of the workings. The progress of subsidence with the advance of the excavated area is shown in Figure 4. As the excavated area is enlarged, the subsidence profile gets deeper, until a critical excavated area is reached and maximum subsidence occurs. Further excavation results in an enlargement of the area of maximum subsidence. Subsidence phases over areas of excavation smaller than this critical area are termed "sub-critical" while those over areas larger than the critical area are termed "super-critical".

The maximum subsidence over a critical or super-critical excavation is empirically considered to be a function of the mining seam thickness,

and is usually related in the following manner:

$$S_m = a \times t$$

where:

S_m = maximum subsidence

a = subsidence factor

t = seam thickness

Table 1 contains subsidence factors for various coalfields, as compiled by Braeuner (11). The subsidence factor is dependent on the mining method, system of packing, and influence of overlying strata condition.

2.1.1.2 Lateral Displacement

Generally the lateral movement of a subsiding point is towards the centre of a mining panel, and zero over the centre of the panel. The horizontal displacements are reported to be appreciably larger than could be expected from only the curvature due to the subsidences (11). The magnitude of the horizontal displacement is found to be roughly proportional to the slope of the subsidence curve.

2.1.1.3 Lateral Strain

Lateral surface strain tends to be compressive within the bounds of the excavation and tensile outside the bounds of the excavation. The transition point between compressive and tensile strain corresponds to the inflection point of the surface displacement curve, which generally tends to be the point of one half maximum subsidence. This point is located inside or outside the excavation boundary depending on the geology of the strata overlying the excavation, and the geometry of the mine opening. The N.C.B. found the position to vary depending on the width/depth ratio of the excavation.

2.1.1.4. Limit Angle (Angle of Draw)

The limit angle is defined as the angle between the horizontal(U.S.) or vertical(U.K.), which joins the edge of the working to the edge of the subsidence area. (For simplicity sake it will be considered to be the horizontal angle throughout this report). Table 2 lists some typical limit angles as reported by Braeuner (11).

The location of the edge of the extent of subsidence is dependent entirely on the accuracy of the survey of that point. Small errors in measured displacement can result in large errors in the calculation of

the limit angle (39). Certain conditions such as uplift at the edge of the workings can further obscure the location of the limit of subsidence.

An important parameter needed as input for use of many of the empirical prediction techniques is the critical radius. This is defined as the radius of the critical area of working which causes the complete subsidence of one point on the surface. It is considered to be independent of the limit angle, however it is often approximated from the limit angle as follows:

$$R = h \times \cot \gamma$$

where:

R = critical radius

h = depth of working

γ = limit angle

2.1.1.5. Break Angle

The break angle is defined as the angle of inclination of the line joining the edge of the workings to the point of maximum tension. This point is generally found to be located outside the limits of the excavation.

3. SURVEY OF SUBSIDENCE PREDICTION METHODS

As mentioned previously, predictive methods can take the form of:

i) Empirical or

ii) Analytical methods.

3.1. EMPIRICAL METHODS

Emprical methods are based largely on experience, intuition and observation of ground movement, without any particular regard to deformable body mechanics. These models can further be broken down into Graphical and Mathematical methods.

3.1.1. GRAPHICAL METHODS

Graphical methods rely on the compilation and summary of case historys in a graphical form, from which predictions of subsidence in areas of similar geology and geometry can be made.

3.1.1.1. National Coal Board Subsidence Engineers Handbook

This handbook was published in 1966 (51), and revised in 1975 (52), as a practical guide for the prediction of surface subsidence above U.K. longwall coal mines. It represents a large collection of subsidence field data from U.K. longwall mines, summarized and analysed through a series of non-dimensionalized graphs. These graphs allow the user to calculate the subsidence for mines of similar design and geology, as those summarized in the handbook, to be predicted.

A series of these charts are shown in Figures 5a,b,c.

Basic requirements are that the mined excavation is rectangular in plan and sufficiently long. The first graph (Fig.5a) predicts the ratio of maximum subsidence to the thickness extracted in terms of the width/depth ratio of the excavation. The use of the graphs assumes the principle of equivalent extraction areas in which mining areas of similar width/depth create similarly shaped subsidence profiles. Maximum subsidence is reduced if the length of the panel is not long enough to be termed critical (Fig.5b). The third graph (Fig.5c) produces a complete subsidence profile across the panel which may be further corrected if the seam is tilting. Graphs are given for the prediction of surface strain and a nomogram is used for estimating the time taken for completion of subsidence once the face passes a point. Corrections for the effects of pillars located between wide panels are also given.

This method is claimed to give reasonably accurate prediction, with the maximum subsidence claimed to be correct to within ten percent "in the great majority of cases".

Limitations of the N.C.B. method

The N.C.B. handbook is intended for prediction of subsidence above U.K. longwall coal mines, with excavations rectangular in plan and of a sufficient length. The angle of draw is assumed to be 55 degrees for the U.K. conditions and it is expected that if the predictive technique is applied to areas of significantly different angle of draw, errors in the shape of the predicted curve will result.

The predictive procedures are based entirely on surface survey data, and no reference is made to geology, topography, material properties of the subsiding rock mass or methods of mining, all of which influence general subsidence development.

3.1.1.2. Graphical Methods for Room and Pillar Geometrys

Attempts have been made to adopt an empirical approach, similar to that proposed by the N.C.B., for prediction of subsidence over room and pillar workings. Plots were developed for first mining (46) as shown in Figures 6,7,8,9, based on limited subsidence data available over room and

pillar coal workings. The subsidence factor was related to:

i) $D \times H/[(1-R) \times W]$

ii) $D \times W \times (1-R)$

iii) $D \times H \times R/W$

iv) $D/(W \times (1-R))$

where:

D = depth of working

W = pillar width

H = seam thickness

R = extraction ratio

It is suggested that such relations should be used with caution as they are based on limited subsidence data.

3.1.2. PROFILE FUNCTIONS

The method of profile functions employs matching the observed subsidence profile with a mathematical function. This technique is used successfully in Poland, Hungary, and the U.S.S.R..

A number of different types of profile functions, representing the

profiles found over critical areas, are shown in table 3. All are of the general form:

$$s = S \times f(R,x,c)$$

where:

s = subsidence at a point

S = subsidence at the centre of the profile

R = critical radius

x = horizontal distance from inflection point

c = either a function or a constant

Profile functions for subcritical areas are shown in table 4. These are derived from the critical profiles by using certain empirical and/or mathematical relationships.

The profile functions are representations of 1/2 of the subsidence profile, ranging from zero subsidence to full subsidence (Fig.10). The location of the subsidence point is given as a distance from the inflection point of the subsidence curve, in terms of the seam depth. In most cases the inflection point is located at the point of one half the maximum subsidence. The location of the point of inflection of the subsidence curve in relation to the edge of the excavation must be derived from field measurements.

Profile functions are either of an asymptotic or non-asymptotic shape (Figs.11,12). The hyperbolic and error profile functions are asymptotic to s=0 and s=S, so theoretically full subsidence and zero subsidence are never found. Calculating the subsidence of a point located at the crit-

ical radius from the inflection point results in subsidence of 98% of the maximum subsidence for the hyperbolic function and 99% for the error function, therefore the difference is insignificant.

Profile Function Limitations

The application of profile functions is generally limited to longwall mining panels of rectangular shape, in which the profile coincides with the centre line of the excavation. The method is not generally applicable to complex mining geometry such as those found in room and pillar mining.

Profile functions are derived from observed subsidences with no explicit dependence on geological factors. Each of the functions pertains to a given district, with unique geological setting.

3.1.3. INFLUENCE FUNCTIONS

The theory of influence functions is based on the concept of superposition. An excavation can be thought of as being composed of an infinite number of infinitesimal elements. The subsidence of a surface point can be thought of as the sum of the influences of all the infinitesimal elements of excavation . This concept is illustrated in Figure 13, where the total subsidence curve is the summation of all the individual elemental subsidence curves. The contribution of each excavated element on

the subsidence of an overlying point is the product of it's area, dA, and the magnitude of the influence that dA has on that surface point. This influence, p, is a function of the radial distance, r, of this extracted area from the surface point when observed in plan view, and is represented by:

$$p = f(r)$$

This is referred to as the influence function. The total subsidence at a point is equal to the total sum of the influence of each element extracted. The extracted element lying directly under the point will have the greatest influence on it's subsidence, while points lying at a distance from the point will have lesser influence (Fig.14).

Each point on surface can be assigned an influence area, of radius equal to the critical radius, where the complete excavation of this area causes full subsidence. Incomplete excavation will cause some portion of maximum subsidence to be realized. For example the point overlying the edge of a mined out excavation (Fig.15), will be influenced by one half of all the possible influences, therefore should theoretically undergo one half of maximum subsidence.

The subsidence of a point can therefore be expressed as the integral of the influence function over the area of influence as follows:

$$S = \int_A p(r)dA$$

Many continuous influence functions have been proposed, from the observations of subsidence profiles in various regions, these are listed

in table 5. All but one of the equations are of the form:

$$p(r) = k_1 \times s(R, r, k_2)$$

where:

$p(r)$ = profile function

s = subsidence at the panel centre

R = critical radius

r = radial distance from the point under consideration

k_1, k_2 = constants

(Equation 1, Table 5, includes the variable - angle of break)

When solving for the subsidence over the edge of a critical or super-critical excavation, one half of the influence area would be excavated, implying one half of maximum subsidence at that point and inflection of the subsidence profile. In many cases the inflection point is found over the excavated area at a distance, d, from the excavation edge (see Fig.10), therefore the profiles determined from the influence functions deviate from observations. This is most often corrected for by deducting a "compensation" zone from the excavation equal in width to the distance d (Fig.16). This compensation zone is treated as if it were not mined.

Manual methods are also used to determine influence functions (45). In this case the influence area of a point is divided into a series of concentric circles or zones, the width of each zone usually a proportion of the seam depth (Fig.17) With this method, the subsidence of a surface

point is represented by the expression:

$$u = Aa+Bb+Cc+Dd+.....+Mm$$

where:

u = subsidence as a proportion of seam thickness

A,B,C.. = proportion of zone extracted

a,b,c.. = influence factor of each zone

If we have m zones and m known subsidences we can assemble m equations, to solve for the values of the m influence factors.

In the case that the location of the point of one half subsidence doesn't coincide with the edge of the workings a power series be used to represent the subsidence at a point, therefore:

$$u = A^n a + B^n b + C^n c + ...+M^n m$$

The value of n can be determined by comparing subsidence at the edge of the excavation to full subsidence, where:

$$S = 1.00^n a + 1.00^n b + 1.00^n c + ...+1.00^n m$$

$$s = 0.50^n a + 0.50^n b + 0.50^n c + ...+0.50^n m$$

where:

S = maximum subsidence

s = subsidence over the edge of the excavation

therefore:

$$n = \ln(s/S)/\ln(0.50)$$

In the case of subcritical excavations the above equation must be

in the following format:

$$n = [\ln(S)-\ln(s)]/[\ln(2.0)-\ln(Ae)]$$

where:

Ae = the proportional extracted area

Effect of Pillars

Empirical studies in the coal fields of U.K. have shown that the recorded subsidence over the centre of a pillar separating two adjacent panels is less than that expected when simply superimposing profile functions, when the pillar width/depth ratio is between 0.1 and 0.5 (Fig.18). Work done by Marr (45) in 1975 utilized a zone area method to determine the subsidence reduction attributed to pillars of width/depth of less than 0.5. Experience in the U.K. showed that subsidence above pillars of W/D ratio of 1.0 was zero, so a maximum reduction radius for the influence of pillars of 0.5×depth was used. The area of influence of the pillars was broken into 5 concentric rings and observation of reduction profiles yielded the following relationships:

$$Z = .215U + .300V + .270W + .110X + .005Y$$

Where:

Z = zone value due to pillars

U,V,W,X,Y = Proportional pillar areas

(valid for W/D down to 0.1)

A correction value is applied to the derived zone value to determine the reduction value. The correction values are based on observed subsidence and shown in Figure 19.

The way reductions are applied are as follows:

i) the total expected subsidence at a point is determined by means of influence functions, this includes the area of the pillar.

ii) the zone value of the contribution of the pillars is determined by a zone area method.

iii) a correction of the zone value is made through Figure 19.

iv) the corrected reduction factor is added to the expected subsidence to arrive at the predicted subsidence.

The reduction formulae, and parameters determined were for pillars with W/D dimension of between 0.1 and 0.5.

Effect of Time

In the presentation of the influence function method by Hiramatsu and Oka (28), the contribution of the mined element to the subsidence of a point is given as follows:

$$s = m \times a \times T(t) \times p(r) \times dA$$

where:

m = thickness of the seam

a = subsidence factor

$T(t)$ = time factor which is a function of the time elapsed from the instant of mining

$p(r)$ = profile function

The time factor would assume a shape similar to that shown in Figure 21, and would range from a value of zero at the time of excavation to one at the time where full subsidence is acheived. When determining the subsidence at a point the time function must be attributed to each mined element influencing that point.

Limitations of the Influence Function Methods

The use of influence functions relies on the theory of superposition being valid. This is equivalent to saying the overburden behaves in a homogeneous, isotropic manner. There is no explicit dependence of the influence functions on the geological factors.

A fundemental assumption of the method is that each elemental extracted area has the same subsidence effect on surface as any other element. This would infer that the extraction of a pillar in a highly excavated area would have the same effect as the excavation of the same volume of ground in an unmined area.

Arbitrary adjustments near the excavation limits, such as compensation zones must be made.

3.1.4. COMPLEMENTARY INFLUENCE FUNCTIONS

The use of complementary influence functions for the prediction of subsidence has been investigated by Sutherland and Munson (75). With this method the subsidence of a surface point is considered to be the result of the combined influence of both mined and unmined areas. Influence functions are attributed to both mined and unmined elements (Fig.20). To determine subsidence at a point the influence of the unmined area and mined areas are integrated over their influence areas and summed to produce a subsidence prediction.

The elemental response of the unmined area is based on the elastic response of a plate supported by an element and given the following formulation:

$$S_s/m = P_s \begin{cases} 0 & , 0<r<b \\ [I][\Delta][r_s^2(1-2\ln(r_s^2)] & ,b<r<a+b \\ 1 & , a+b<r \end{cases}$$

where:

r_s = $(r/a)-(b/a)$

S_s = vertical subsidence around unmined element

m = mined height

P_s = proportion of maximum subsidence attributed to unmined element

r = radial coordinate from centre of element

b = half width of element

a = radial extent of influence outside the element

[I] = effect of moment of inertia of overlying strata

[Δ] = effect of crushing of unmined material on element response.

Physical and numerical models of the response of strata over a longwall panel indicates a thinning of the elastic beam as one moves away from the rib. The elastic solution can be modified to take into account this thinning, through the moment of inertia function:

$$[I] = [t_0/t(r_s)]^3$$

where:

$t(r_s)$ = thickness of plate as a function of radius

t_0 = centre thickness of plate

In analysing the response of the mined element an error function, based on the probability of the migration of voids through caved overburden(71), was used:

$$S_v/m = P_v \begin{cases} 1 & , 0<r<b \\ \mathrm{erfc}(2r_v)=1-2\sqrt{\pi}\exp(-\xi^2)d & , b<r<c+b \\ 0 & , c+b<r \end{cases}$$

where:

$r_v = (r/c)-(r/b)$

S = vertical subsidence due to mined element

P = proportion of maximum subsidence due to mined element

c = radial extent of influence outside the element

ξ = integration parameter

In the case of a longwall panel where:

a,c = one half the range of the draw angle

P_s = related to the center line displacement

P_v = related to displacement over the ribside

The predicted results agreed quite closely to field measurements.

The use of complementary influence functions has seen little practical application, however it overcomes some of the limitations of the influence function method, particulary the problem of compensation zones.

3.1.5. INCREMENTAL DESIGN

The prediction of subsidence by incremental design techniques has been formulated by Coates (18), primarily for use within the U.K. coal fields. The basis of the method is that measurements of subsidence above mining excavations reflect the effect of mechanical properties of the ground and the mechanisms that govern deformation. Using initial measurements as a base, predictions can be made of the subsidence to be caused by enlarging the mining area and prediction of subsidence can be extrapolated to areas of similar mining geometry and geology.

The calculation of maximum subsidence above a mining panel, as

determined from U.K. coal mining experience, is as follows:

i) for subcritical excavation , $Y/Z, L/Z<1.4$

$$V_m/H = q \times n(Y/Z(2.8-Y/Z)^{1.5}$$

and

$$n = 0.365(L/Z(2.8-L/Z)^{1.5}$$

ii) for critical-supercritical excavation , $Y/Z, L/Z>1.4$

$$V_m/H = 2.74q$$

where:

V_m = maximum settlement

H = mining seam height

q = parameter varying with overburden rock and type of support

L = panel length

Y = panel width

Z = panel depth

For incremental design:

i) for subcritical excavations

$$V_m^b/V_m^a = ((L_b/L_a)(Y_b/Y_a)(B_b/B_a))(A_b/A_a))^{1.5}(Z_a/Z_b)^3(H_b/H_a)$$

where:

subscript 'a' refers to previous cases

subscript 'b' refers to future cases

ii) for critical-supercritical excavations

$$V_m^b/V_m^a = ((Y_b/Y_a)(Z_a/Z_b)(A_b/A_a))^{1.5}(H_a/H_b)$$

An empirical equation giving approximate shapes of the transverse profile in the area where V/V_m is less than 0.5, is given below. This

area is usually where horizontal strains, tilt and differential vertical settlement start to cause damage to structures.

$$\lg(V_m/V)=(f\times L/Z+0.03)(x/x_s)^2$$

where:

f = a constant for the site

x = horizontal distance from panel centerline to point where V to be predicted

x_s = horizontal distance from centerline to point where V=1/2 maximum V

For incremental purposes:

$$C_b=((C_a(x_s/x)_a{}^2-0.03(Z_a/L_a)(L_b/Z_b)+0.03)(x/x_s)_b{}^2$$

where:

$$C = \lg(V_m/V)$$

or:

$$(x_s/x)_b{}^2=((C(x_s/x)_a{}^2-0.03(Z_a/Z_b)(L_b/L_a)+0.03)(1/C_b)$$

To determine x_s for any particular site, field measurements can be applied to the following empirical equation, for $0.5<L/Z<2.4$:

$$x_s/Z = 0.37(L/Z-0.5)^{0.5}+d$$

where:

d = a parameter depending on rock properties, mining method, degree of support, L/Z, V_m/V

For incremental design purposes, the factor d can be eliminated as follows:

$$(x_s)_b/Z_b = (x_s)_a/Z_a - 0.37(L_a/Z_a - 0.5)^{0.5} - (L_b/Z_b - 0.5)^{0.5}$$

or:

$$(x_s)_b/Z_b = (x_s)_a/Z_a - 0.37(F_a - F_b)$$

where:

$F = 0.5$,for $L/Z<0.5$

$F = (L/Z-0.5)^{0.5}$,for $0.5<L/Z<2.4$

$F = 2.4$,for $L/Z>2.4$

The tilt, or the slope of the subsidence curve can be determined from the theoretical relationship $t=dV/dx$. From the formula for the subsidence curve, tilt may be expressed as:

$$t = 4.60 \times C \times V/X$$

where:

$$C = (f \times L/Z + 0.03)(x/x_s)^2$$

In incremental design form:

$$t_b = t_a(C_b/C_a)(V_b/V_a)(x_a/x_b)$$

The horizontal displacement, due to bending of the strata, has shown by laboratory experiments, to be related to tilt in the following form:

$$X = Z \times t$$

where:

X = horizontal displacement

Z = depth of strata

t = tilt

When the depth is greater than half of the span L, much of the horizontal displacement is due to shear strain. Horizontal friction between rock strata further effects the amount of horizontal displacement. Therefore the formula for horizontal displacement is modified to:

$$X = g \times Z \times t$$

where:

g = a constant for a particular rock and mining method

In incremental form:

$$X_b = X_a(Z_b/Z_a)(t_b/t_a)$$

The calculation of horizontal strain, e, is determined from the horizontal displacement as follows, $e = dX/dx$, which in effect requires the double differential of vertical displacement. It is advised by Coates, that strain be determined by calculating the horizontal displacement at two points bracketing the point in question and dividing the difference in displacements, by the difference in location. In effect determining, $(\Delta X/\Delta x)$. It is pointed out that the strain calculation is inherently quite inaccurate as it requires several empirical extrapolations and is based on two differences.

Limitations of the Incremental Design Method

The relations between the geometry of the mining method and surface movement have been determined from conditions in the U.K.. If these relations are used in areas of different geology and mining methods, the accuracy of the predictions could be quite low.

The techniques of incremental design could be applied to other areas as long as the parameters applicable to the site have been identified. This method does not take into account the effects of time on the various ground displacements.

3.2. ANALYTICAL PREDICTIVE METHODS

The analytical predictive methods are based on the physical properties of subsiding materials, and their response to the stresses induced by the mining geometry. Various approaches have been taken towards the analytical prediction of subsidence, including closed form elastic formulation and numerical methods.

3.2.1. CLOSED FORM ELASTIC FORMULATION

On the basis of elastic theory, Berry (8) considered the deformation caused by a thin slit on a semi infinite, transversely isotropic plate. The planes of material symmetry were considered to be parallel and perpendicular to the ground surface. This configuration could simulate a tabular type orebody in a stratified rock (Fig.22).

The resulting expression for subsidence on surface and the resulting

horizontal strain, in two dimensional space, are as follows:

$$v_0(x) = -\frac{t}{\pi(\alpha_1-\alpha_2)}\left\{\alpha_1 \tan^{-1}\frac{2ah_1}{x^2-a^2+h_1^2} - \alpha_2 \tan^{-1}\frac{2ah_2}{x^2-a^2+h_2^2}\right\}$$

$$\varepsilon_0(x) = \frac{2a\alpha_1\alpha_2 t}{\pi(\alpha_1-\alpha_2)}\left\{\frac{x^2-a^2-h_1^2}{(x^2-a^2-h_1^2)^2+4h_1^2x^2} - \frac{x^2-a^2-h_2^2}{(x^2-a^2-h_2^2)^2+4h_2^2x^2}\right\}$$

where:

a = half width of the opening

h = depth of the opening

t = vertical roof convergence

$h = h/\alpha_i$ (i=1,2)

are the transverse isopropic elastic parameters

and α_1, α_2, are the real roots of the

equation $\alpha^4-2K_2\alpha^2+K_1^2 = 0$

where:

$K_1^2 = (1-\nu)/(E_x/E_y-\nu_y^2)$

$K_2 = \{0.5E_x/M-\nu_y(1+\nu_x)\}/(E_x/E_y-\nu_y^2)$

and G = the shear modulus in the vertical plane

It is difficult to derive the elastic constants for the property of the composite material overlying a mined orebody, however, Berry has derived a method of approximating constants K_1, K_2, from the shape of the subsidence profile. In cases where the W/D ratio of a horizontal excavation is small, and there are no previous excavations in the vicinity

to affect ground movement, the slope of the subsidence curve at the point of 1/2 maximum subsidence may be used to derive the constants as follows:

$$K_1 = \frac{h^2}{x_1^2}\left[2x\frac{dV}{dx} - 1\right]_{x=x_1}^{0.5}$$

$$K_2 = \frac{h^2}{x_1^2}\left[1 - x\frac{dV}{dx}\right]_{x=x_1}$$

where:

x = distance from the center of the panel to the point of 1/2 max. subsidence

dV/dx = slope of the point at x, where V has been normalized such that maximum subsidence = 1.0

Having obtained constants K_1, K_2, a subsidence curve can be plotted and compared to actual measurements. A good fit would indicate the theory gives a reasonable representation of the material properties, and the method of determining the factors K_1, K_2. A poor fit would indicate the theory is not applicable or the estimation of the factors K_1,K_2 are not good enough.

Analysis for cases of non-closure, partial closure, and three-dimensional treatment of a panel is included in appendix 2.

Limitations of the Closed form Elastic Analysis

A very simple idealization of the mining geometry is assumed with this method, as well as a simplified assumption of the properties of the material representing the overburden. For this reason this method should be limited to use with regular shaped excavations.

This elastic analysis has had some success in approximating observed subsidence above some U.K. longwall coal mines.

3.2.2. GENERALIZED INFLUENCE FACE ELEMENTS

The concept of face-elements is much the same idea as the influence function concept, however the displacement at a point is based on elastic analysis. To determine the influence an individual face element has on a point, it's distance, area, and average convergence must be known. By substituting these values into the appropriate expression of the elementary vertical displacement, the contribution of each face element to the total vertical displacement at a point P can be determined. Total vertical displacement can be determined by summing the elementary contributions. The same procedure can be used for the calculation of any other displacement, strain or stress component.

For the configuration shown in Figure 23, the influence of a single extracted area, ΔA, on the displacement of the point P can be given as:

$$w = \frac{S_z}{\pi} \frac{\Delta A}{z^2} F_1$$

$$u = -S_z \frac{r}{z} \frac{\Delta A}{z^2} F_2$$

where:

w = vertical movement

u = displacement in radial direction

S = convergence of the face element

ΔA = area of face element

z = vertical distance from free surface to point

F_1, F_2 = function depending on mechanical properties

For the case of a homogeneous isotropic model:

$$F_{1*} = \frac{z^2}{4(1-\nu)} \left\{ \frac{2(1-\nu)\, z_2}{(r^2+z_2^2)^{3/2}} + \left[z + 2H(1-\nu) \right] \frac{2z_2^2 - r^2}{(r^2+z_2^2)^{5/2}} + 3Hzz_2 \frac{2z_2^2 - 3r^2}{(r^2+z_2^2)^{7/2}} \right.$$

$$F_{2*} = \frac{z^3}{4(1-\nu)} \left[\frac{1-2\nu}{(r^2+z_2^2)^{3/2}} - \frac{3z_2(z_1 + 2\nu H)}{(r^2+z_2^2)^{5/2}} + \frac{3Hz(4z_2^2 - r^2)}{(r^2+z_2^2)^{7/2}} \right]$$

where:

$z = H$ = depth of seam

ν = Poissons ratio of material

This elastic analysis can be extended to the case of the frictionless, laminated model, homogeneous isotropic medium, homogeneous transversely

isotropic, and multi-membrane models. Derivations of the functions are included in Appendix 3.

Salamon (61) states the face element principle can be applied if three major requirements are met; 1) The mathematical model which describes adequately the behavior of the rock mass surrounding the seam must be determined. This model can be acheived by "meticulous" analysis of the results of in-situ experiments. 2) The magnitude and distribution of convergence of roof and floor must be known to employ the face element principle. Furthermore this must be the elastic portion of the actual closure. 3) Displacement, stress and strain components at any point in the solid rock mass may be determined at any point in the rock mass by evaluating definite integrals. Except in the case of simple geometry, these cannot be determined by exact mathematics, so a numerical integration method must be employed.

Limitations of the Face Element Method

The application of the face element method is dependent on the above three conditions being met. In practice these conditions are difficult to acheive. Large scale in-situ tests are very rarely done on a rock mass, and even then the results can't be guaranteed to adequately represent the characteristics of the entire rock mass. Recording of subsurface subsidence data is an extensive operation, and rarely continuous, as access to certain areas of the mine changes continuously due to advance of the working face and ground condition. Distinguishing the component of elastic ground closure from the total movement is difficult if not

impossible. The complexity of the mathematics of the method require numerical integration techniques, which require the availability of a computer.

3.2.3. NUMERICAL METHODS

In more recent analysis (15,23,25,40,41,48,59,69,71,72,78,86) numerical computer techniques have been used to predict subsidence. With these techniques the behavior of overburden strata, and ore remaining after mining, in response to the stress fields induced by the geometry of the mining, are analysed. Calculations of great detail are possible with these methods, and have shown how elastic bending, breaking, bulking, and void volume are transmitted through the ground mass to cause subsidence.

3.2.3.1. Void Volume Methods

A numerical method of subsidence prediction, in the form of a void volume model has been presented by Munson and Benzely (48). It is based on the assumption that three basic failure modes dominate mine subsidence. The three modes consist of progressive roof failure, progressive pillar failure and time dependent deformation of recompaction.

A simple bulking model is proposed that allows significant amounts of volume to propagate from the mine workings to surface. The governing

equation takes the form:

$$S = a \times V$$

where:

S = subsidence volume

a = subsidence parameter

V = the mined volume

The subsidence parameter contains all material and geometry aspects that control subsidence.

It is envisioned that pillars and roof of a mined area deteriorate with time and eventually fail. As the roof blocks fail and fall into the mined area they occupy greater volume in a way that depends on their size and geometry in relation to the room volume. Subsequent falls of ground will have less volume to occupy. The bulking will change with time as compaction takes effect due to the weight of the overburden. This is

incorporated into the subsidence factor as follows:

$$a = a_0 + a_1(\sigma(t), M(t) + a_2((t), M(t))$$

and

$$t_0 = t_0(\sigma(t), M) \quad , t > t$$

where:

a = time independent bulking factor

a_1, a_2 = time dependent material behavior

t = time

σ = stress

M = time dependent material behavior

t_0 = the initial time which is unique for each individual roof layer.

These modes of material response have been tested by means of numerical methods, and compared to results obtained from small scale physical models, and full scale longwall panels. Results obtained have had varying agreement with observations.

Limitations of the Void-Volume Method

There is a difficulty in predicting the failure sequence of mine strata, and determining the physical parameters of the individual layers.

3.2.3.2. Finite Element

Most of the interest shown recently in the application of mechanics to the prediction of subsidence has been concentrated on finite element analysis.

In this method the rock mass is modelled mathematically as a finite number of idealized elements, in either two or three dimensions (Fig.24). Each element deforms according to a characteristic stress-strain relationship, while retaining equilibrium and compatibility between elements. The rock mass may therefore be represented by many different layers, or regions. Discontinuities, complicated mining geometry and surface geometry may also be incorporated into the model. Through mathematical analysis the displacements, strains and stress fields throughout the rock mass may be solved for.

The techniques used for solving the finite element method are very sophisticated and the method has a literature of it's own. There are numerous texts dealing with the science of finite element analysis, a list of some of them is included in Appendix 4. There is a wide availability of user oriented programs available on the market.

The practical use of finite element methods should be preceeded by a comprehensive study of the field measurements of displacement, stress and material properties of the rock mass. These parameters are required as input for the method.

Limitations of the Finite Element Method

Since there are a large number of algebraic equations to solve with this method, involving the input of many field parameters, it is quite easy to retrofit initial parameters to suit observed displacements. Similar predictions of displacements at surface could arrive from numerous combinations of these initial parameters. In order to assess the credibility of derived results, a rigorous assessment of the material properties, and boundary conditions making up the rock mass must be made before the method is used.

Unlike many engineering problems, ground movement and stresses involve large regions of area and there is difficulty in determining the location of the boundary inside of which all significant events will occur. The modelling of a mining environment involves vast amounts of data, the capacity of the computing facility, or the cost involved in the analysis may limit the detail to which it may be represented.

3.3. TIME DEPENDENCE

Most observations indicate that subsidence is a time dependent phenomenon. Subsidence can start from days to weeks after the onset of mining and continue for months to years after it's completion. The time it takes for a point on surface to subside to it's maximum displacement will depend on the time it takes for the critical area to be excavated, which in turn will depend on the rate, and depth of mining.

Braeuner (11) has summarized four other factors effecting the duration of movement in longwall mining; 1)Strata condition - it has been found that the subsidence process lasts longer in cases of stiff, competent overburden strata, than in the case of thin, compliant strata. 2)Depth of extraction - generally the movements will start later and last longer over a deep excavation as compared to a shallow excavation. 3)Type of packing - in the case of packing, or fill, the duration of movement is longer than in the case of caving, however the magnitude of movement is less. This is more than likely to do with the recompaction of the fill. 4)Former excavations - strata that has already been deformed by previous workings, tends to react at a greater rate than virgin ground.

An analytical time factor expression is commonly used to relate the

proportion of full subsidence to time. It is as follows:

$$s/S = T = [1-\exp(-ct)]$$

where:

s = subsidence at time t

S = maximum subsidence

c = a site constant obtained by field observation

In room and pillar mining the rate and duration of subsidence is related to the long and short term strength and deformation characteristics of the pillars, roof amd floor. Extraction ratio, stress levels, and pillar geometry, all have an effect on these characteristics.

Creep curves for model potash pillars, derived by King (41), show a geometric relationship between the vertical strain on the pillar and time, of the form:

$$e = a \times t^b$$

where:

e = material strain in vertical direction

t = time

a,b = constants depending on the material

In room and pillar mining the time factor expression could be expected to be a combination of the time effects of both the overburden and pillar response.

4. APPLICATION OF EXISTING SUBSIDENCE PREDITION METHODS TO THE CASE OF SASKATCHEWAN POTASH MINES

Most empirically derived predictive methods have been developed for use with European long wall coal mines. The direct application of these techniques to the Saskatchewan potash mining case is not possible due to the differences in mining method and properties of the mined material and overburden. The combination of room and pillar type of mining and complicated mining geometry employed by the Saskatchewan Potash mines will preclude direct application of many of the methods developed for use with regular geometry longwall mining panels. The tendency of potash pillars to creep plastically, makes it necessary for the effects of time to be taken into account.

Spread sheets summarizing and comparing the various methods in a point form are included in appendix 5.

Generally the N.C.B.'s graphical technique has had little success in approximating observed results when applied to areas outside of the U.K. (49,38)

Due to the complicated geometry of the mining plans of the potash mines, a continuous profile function will not be able to accurately represent the shape of the subsidence profile. The shape of the profile function is derived from the measured profile of a critical exavation

which does not occur in the Saskatchewan mining case due to the room and pillar method employed.

The concepts of influence functions or complementary influence functions have definite advantages for their application as a predictive technique for room and pillar potash mines as they can be used with complicated mining geometrys and allow application of a time factor. The zone area method of subsidence prediction is investigated in this report for application as a predictive technique.

The concept of incremental design may prove to be applicable to the potash mining situation once sufficient number observations of the subsidence development has been made to determine the factors effecting subsidence.

The analytical/mechanistic techniques of subsidence prediction may prove to be applicable, once a history of the physical properties of the overburden has been collected, and the mechanics of subsidence have been determined.

The results of a study carried out to test the applicability of a packaged subsidence prediction program is included in Appendix 6. The method is based on a finite element program, using simple beam elements to represent the overburden.

4.1 DATA RECEIVED FROM THE SASKATCHEWAN POTASH MINES

In order to develop and test a subsidence prediction technique, a data base of subsidence information from the Saskatchewan potash mines was required. A subsidence questionnaire was distributed to five producing potash mines, requesting information concerning their current subsidence survey technique and practice, measured results, and geometry of the mine workings. Information was received from Allan, Central Canada Potash (CCP), Cory, Lanigan, and Rocanville mines.

A copy of the subsidence questionnaire is included in Appendix #7.

4.1.1. CURRENT MINING PRACTICE IN THE SASKATCHEWAN POTASH MINES

In North Saskatchewan, the potash ore is predominately excavated from the upper zone of the Patience Lake member of the Prairie Evaporite formation, more than 1000 metres below surface. The Prairie Evaporite is composed of alternating beds of halite and sylvite, having an average thickness of approximately 200 meters. At CCP and the PCS Mining Allan and Cory Divisions, the main ore horizon is situated near the top of this formation, however, due to the requirement for improved long term stability of the main entry for ore handling, major underground arteries

have been developed in the ore zone below the existing workings. At PCS Lanigan the main ore zone is situated at this lower horizon. At Rocanville, the lowest potash member (Esterhazy) is being mined.

Two main continuous mining layouts have evolved in the Saskatchewan Potash mines; 1) Pillar Panel mining (Fig.25) and 2) Chevron mining (Fig.26). The method utilized at any particular mine, and associated room dimensions are functions of the ore bed type and ground stability. A review of the current underground mining methods used in Saskatchewan Potash is provided by Jones and Prugger (36).

4.1.2. CURRENT SUBSIDENCE SURVEY PRACTICE

4.1.2.1. Allan

The Allan subsidence survey circuit contains 175 stations installed between 1968 and 1978 (Fig.27). They are generally located in North/South and East/West lines, along section and quarter section boundaries. The spacing between the N/S stations is approximately 1/4 mile, while those in the E/W direction are at 1/2 mile intervals.

The mine workings run in N/S and E/W direction, and are effectively bisected by the lines of the established grid. The current workings located to the west of the shaft are chevron rooms running in a N/S direction. Two lines of survey stations have been located above these workings, running in an E/W direction, with stations located at 1000 foot intervals. One line is located above the main access drift, the second runs parallel to the first one half mile to the south.

Until 1978, subsidence surveys were balanced with reference to a line of stations lying approximately 1/2 mile north of the shaft located in a zone of active subsidence. From 1978 on, the reference bench mark was located in the town of Allan, approximately 2.5 miles to the south of the workings, well out of the range of influence of subsidence.

Subsidence surveys have been carried out yearly from 1968. Primary survey circuits are run from the reference bench mark to lines extending across the north and south boundaries of the workings, to a first order degree of accuracy. Secondary circuits are run between the two boundary lines to a third order degree of accuracy.

4.1.2.2. Central Canada Potash

The CCP subsidence survey circuit contains 113 stations, installed between 1973 and 1982 (Fig.28). In 1973, 28 stations were installed on

a loose pattern covering Block I and part of Block II. Stations were installed along section boundaries at one mile spacings in the E/W direction and 1/2 mile spacing in the N/S direction. In 1978, 65 stations were installed in 3 major lines. One line extends in an E/W direction above the main access drifts, overlying the approximate centre of the Block I workings. Station spacing ranges between 600 and 1400 feet. A second line extends in an E/W direction across the approximate centre of Block II, overlying the Block II main access drifts. Spacing of these stations ranges ranges between 550 and 750 feet. A third line runs in a N/S direction along a 19000 foot line extending through the centres of Blocks I and II. Spacing of these stations range between 500 and 1400 feet. In 1982, 20 stations were established over block III. These were installed along 4 E/W running section and quarter section lineswith stations located at approximately 1/4 mile spacing.

Subsidence surveys are performed on a yearly basis in the month of December. Since 1973, surveys have been carried out to a third order degree of precision. In 1979 the survey of a central loop of stations was carried out to a first order degree of precision, with outlying loops surveyed to a third order degree of precision.

Various reference bench marks have been used over the years. From 1973 to 1978 geodetic bench mark GBM-475C was used until it was determined that it was located in an active area. In 1979, BM-19 was chosen as a reference bench mark, which was located only 2230 feet from the south end of Block I. This could be considered to lie within an active zone. In 1980, BM-29 was chosen as a reference bench mark, which was located

approximately 5000' north of the northern boundary of Block II and III, and is presently 6300 feet from the nearest working.

4.1.2.3. Cory

Since 1973, 160 stations have been installed for the Cory subsidence survey (Fig.29). The general arrangement is to locate the survey stations along lines running N/S and E/W along section boundaries, at 1/4 mile spacing. A line of stations installed in 1977 runs E/W along a CNR line and bisects a series of S/N driven panels at the north end of the property. The stations in this line have a variable spacing of between 200 and 1300 feet.

The general layout of the mine has main access drifts running in an E/W direction from which panels employing chevron cut rooms are driven north and south. The grid of survey stations, and the lines created effectively cross the workings in directions parallel and perpendicular to the advance of the panels. In a few instances where the grid has proven too coarse for the pattern of mining, additional E/W lines have been introduced along quarter section boundaries.

Subsidence surveys have been carried out on a yearly basis in the fall of each year. Until 1977 the survey was carried out to a second order

degree of accuracy. From 1978 until present day surveys have been carried out to a first order degree of accuracy.

From 1975-77 the reference bench mark was located at a point within the shaft pillar, which proved to be unreliable. A bench mark was subsequently established at the collar of the #1 shaft.

4.1.2.4. Lanigan

The Lanigan subsidence survey presently consists of 158 stations, with 78 installed in 1978-79 (Fig.30) and 80 installed in 1980. The survey stations installed in 1978-79 are located along three main lines. The major line runs in a NW/SE direction and runs through the centre of the room and pillar panels presently being mined. The line runs in a direction parallel to the orientation of the rooms, and perpendicular to the direction of advance of each panel. The major line extends for a distance of 28000 feet and consists of 37 stations with spacing between stations ranging between 500 and 1000 feet. Two secondary lines intersect this major line at right angles. The northern of the two secondary lines runs SW/NE, overlying a panel access drift heading in that direction. This line consists of 10 stations spaced at 1000 foot intervals. The southern of the secondary lines runs through the centre of one of the main panels presently being mined. This line contains 17 stations at 500 foot spacing.

The stations installed in 1980 are located to the south of the present workings, and are located along section boundaries at 1000 foot spacing.

Subsidence surveys have been carried out yearly since 1978 to a first order degree of accuracy. The reference bench mark is located north of the shaft area and lies at a distance of 4000 feet from the main workings. Although the bench mark should be out of the range of subsidence, it may be influenced by uplift due to the flexure and bending of the strata overlying the workings.

4.1.2.5. Rocanville

The subsidence survey at Rocanville consists of 78 stations, installed between 1973 and 1983 (Fig.31). Six survey stations were installed in 1973 in a random pattern above panels 401 and 402. Thirteen stations followed in 1975, being located along section lines overlying the panel. After development and mining of the areas to the south and north of the shaft had been started, an additional 31 stations were installed to monitor above panels 405, 406, and the southern panel. Station spacing varies between 400 and 1000 feet along these lines.

The frequency and accuracy of the subsidence surveys varied due to

the following reasons:

- the 1976 survey was not considered to be reliable by the mine staff.
- there were no surveys for the years 1977, 1978, or 1980.
- the reference bench mark was found to be loose in the ground for the 1981 and 1982 surveys.
- the 1981 survey was stopped midway during November of that year and continued in June of 1982.
- there was no fall survey for 1982.

All of the surveys were carried out to a first order degree of precision.

From 1972 to 1982 the reference bench mark (DNR-4) was located approximately 8000 feet NW of the shaft and 3000 feet from the main workings. This position could be considered to be within the range of subsidence influence. The 1983 survey assumed a new reference point (BM-51), located approximately 4000 feet from the edge of the workings.

4.2. RECOMMENDATIONS FOR FUTURE SUBSIDENCE SURVEY

From observation of the subsidence survey practice currently undertaken, the following points are recommended to be followed to provide the

mine operator with an effective monitoring system, and a reliable data base for future subsidence research and prediction.

1) Survey station installation should precede mining activity whenever possible. This will assure that the complete history of the subsidence process will be recorded, which will provide valuable information for future research.

2) Survey station lines should coincide with the direction of mining, and be located both parallel and perpendicular to the mining panels if possible. Whenever possible lines should be located along the centre line of a panel. This will facilitate ease of analysis when comparing subsidence to the mining activity, as well as provide the operator with the maximum values of subsidence, slope, and curvature of the subsiding ground.

3) Survey lines should extend beyond the edge of the mine workings. The effects of subsidence always carry beyond the edge of the workings, and often uplift is also experienced. Survey lines should be extended beyond the workings to a distance at least equal to the depth of the workings.

4) Yearly subsidence surveys should be undertaken. A regular set of subsidence surveys will indicate trends and characteristics of the subsidence common to the property. Mining plans of the workings at the time of the subsidence survey should be constructed and stored with the survey, for comparison.

5) Subsidence surveys to a first order degree of precision are scientifically desirable. Due to the large distances traversed and the relatively small values of yearly subsidence, the actual subsidence could be lost in error if a lower degree of precision were used. The cost of such survey precision however may not be economically justified. This decision must be taken by the individual operator.

6) The location of the reference bench mark must lie outside the area of the influence of the subsidence process. Observation of subsidence data for Saskatchewan potash mines indicates that subsidence may extend to a distance equal to the depth of workings, from the edge of the workings. Some data indicates that uplift beyond the edge of the workings may occur. For these reasons the reference bench marks should be established at a distance of at least twice the depth of the workings, outside the edge of the workings. This will equate to a minimum distance of approximately 7000 feet. The bench mark must be a properly constructed monument, free from the effects of frost heave.

7) Whenever possible subsurface survey stations should be established in the mine workings below a subsidence line. This could provide valuable information on how surface subsidence corresponds to subsurface closure.

8) None of the mines presently employ a regular horizontal survey. From observation of the surveyed data it is evident that the strains and curvatures caused by subsidence are extremely small, due to small amount of subsidence over great distances. This is reflected in the fact that no mine has reported any damage to structures caused by subsidence. It

would be scientifically desirable to perform horizontal surveys, to record lateral as well as vertical movement of subsiding stations. This would provide a greater understanding of the subsiding material behavior, however, the additional cost of such a survey may not be economically justified. This decision must be taken by the individual operator.

4.3. CURRENT DESIGN OF SUBSIDENCE SURVEY BENCH MARKS

Drawings of the various designs of subsidence surface stations, used by the potash mines are shown in Appendix #8. They are all of the same basic design.

The monuments consist of an 8 inch dia. or greater hole bored at a predetermined location, to a depth of 12 feet of more. The hole is lined with sono tube or equivalent to prevent caving. A steel survey rod is placed into the centre of the hole and set into an anchor of 1 to 3 feet of cement. A plastic or steel pipe is placed over the survey rod and the area between the pipe and sono tube backfilled. The void between the steel rod and pipe is either left empty or filled with old oil or a

bentonite slurry, and the pipe is capped.

4.4. RECOMMENDATIONS FOR DESIGN OF SUBSIDENCE MONUMENTS

All of the monument designs currently used by the Saskatchewan potash mines should do an adequate job and provide years of service as long as certain points are followed.

The most important consideration is to place the anchor of the survey rod at sufficient depth so that it is immune to the effects of frost heave. It is imperative that the top of the anchor holding the survey rod is located below the maximum expected local frostline.

The lining preventing the hole from collapsing should not be set into the anchor in case diplacement of the top of the tube by frost action is transfered to the anchor and survey rod.

A good practice employed by some of the mines is to make sure the survey rod and sleeve pipe are located below the surface of the ground. This will prevent their damage by accidental contact with vehicles. A marker can be placed to the side of the bench mark to indicate it's location , but enough distance should be be left to prevent damage to the monument by the buried end of the marker if it is toppled.

The space between the survey rod and sleeve should be filled with old oil or an oil/bentonite mixture to prevent corrosion of the rod and sleeve.

5. THE DEVELOPMENT OF A SUBSIDENCE PREDICTION TECHNIQUE

5.1. APPLICABILITY OF THE ZONE AREA METHOD

The influence function technique of subsidence prediction was chosen for investigation over the other empirical techniques because it allows for the effect of time to be incorporated into the method and it can be used with complicated mine geometries.

The zone area method of subsidence prediction is a simplified version of the influence function method. The influence function method uses a continuous influence function to represent the effect an extracted element has on the subsidence of a point and integrates this influence over all extracted elements in the zone of influence to determine subsidence. The zone area method divides the zone of influence into a series of concentric rings, each ring assigned a particular weighting value. The extraction within each ring is determined and multiplied by the weight assigned to the ring. The sum of the product from each ring is used to determine subsidence. By using the zone area method the subsidence at a

point can be represented as follows.

$$S = f\ (a_1A_1+a_2A_2+a_3A_3+.....+a_nA_n)$$

where: S = subsidence

a_1-a_n = weight of ring 1→n

A_1-A_n = proportion of ring 1→n extracted

5.2. CHOICE OF DATA SELECTED TO INVESTIGATE THE ZONE AREA METHOD

From the five data sets received in response to the subsidence questionairre, three lines of subsidence surveys were chosen to test the applicability of the zone area method to subsidence prediction. Lines from the CCP, Cory and Lanigan mines were chosen due to the completeness of data and layout of the surveys.

Rocanville surveys were not used due to the scarcity of data. Unreliability of survey data, lack of surveys between 1977 and 1983, and questionable location of the reference benchmark excluded use of the data.

Allan data could not be considered reliable for surveys performed prior to 1978 due to the use of reference bench marks located within the workings. Plans indicating the sequence of mining during the years 1977 to 1981, and information concerning room and pillar dimensions and extraction ratio's were not received in time for use with the surveys.

The survey line used from the CCP data is a 13000 foot line running E/W, located approximately above the access drift to the Block II panels (Fig.32). The line consists of 21 stations spaced between 550 and 750 feet apart, which were installed and first surveyed in 1978, and have been surveyed yearly since. Development of CCP Block II began in 1975, and mining began in 1977. Mining beneath the line is by the chevron panel method, acheiving a room extraction in the order of 60% with an overall panel extraction of 35%.

The line considered from Cory mine consists of 18 stations located along a 16000 foot line, overlying the #2 Panels (Fig.33). The line is located along a CNR rail line running E/W across the property, with station spacing between 200 and 1300 feet. The survey stations were installed and first read in 1977, and have been read yearly since. In 1981 and 1982 the two stations (#20,#19) lying in the centre of the most active area were destroyed and never replaced. Mining commenced under the line in 1976, using chevron panels. The panels acheived an extraction ratio of approximately 64%, while the overall panel extraction was in the 38-42% range. Mining is progressing in an advancing mode, away from the shaft, in an east to west direction.

From the Lanigan data the 'A' line consisting of 17 station, running in a NE to SW direction across the centre panel, was chosen (Fig.34). The line extends for 8000 feet, with even spacing between stations of 500 feet. The line was installed in 1980, and surveys have been taken yearly since. Mining being carried out under the line is by the pillar panel method, with extraction ratios of between 34 and 46%. Mining is in a

retreat mode, towards the main access drift, progressing in a south easterly direction.

All three of the lines are aligned in the direction of the advance of the mining.

5.3. ANALYSIS OF THE ZONE AREA METHOD.

The first step in the application of the zone area method was to determine the size of the area of influence of subsidence. Table #6 lists observations made from the data received, of the angle between the horizontal and the line connecting the edge of the workings to the point of zero subsidence. The range observed was between 46 and 68 degrees, however some of the survey data indicated uplift outside the edge of the workings, which generally resulted in a large observed angle.

From the above findings the area of influence contributing to the subsidence of a point was assumed to be a circle of radius equal to the mining depth. This circle was further divided into 7 equally spaced concentric rings, to represent the zones of influence.

Observation of CCP data indicated that once mining had advanced beyond the zone of influence of a surface station, that station tended to subside at a fairly constant rate. Figure 35 shows the subsidence with time of

survey stations located above Block II, where mining had finished within the range of influence by 1979. The curves show remarkably constant subsidence rates.

This constant subsidence could be attributed to the creep of the salt pillars in response to the increased stress caused by excavation, and the degree of restraint removed by the mining of the rooms. This is supported by Baar's (4) observation at IMC's mines, that immediately after an opening is made there is rapid convergence of the opening, which decreases until after a 100-200 day interval it settles to a constant rate (Fig.36).

This leads to the theory that subsidence above a potash mine is caused in two stages. Upon mining of a room one contributing factor towards subsidence at surface is the initial convergence of the back and floor of the room into the opening. This displaced volume eventually propagates to surface as one component of subsidence. The second contributing factor is the flow of the salt pillars due to the superincumbent loading and the removed restraint caused by mining. Salt rock is known to have elastic strength in the order of hundreds of p.s.i., while the original superincumbent load at this depth is in the order of 3400 p.s.i.. The pillars cannot support the weight of this load and therefore creep towards the openings.

Comparison between the constant subsidence rates of the monuments above CCP Block II and the extraction ratio within the range of influence of the monuments indicate a close linear relationship between extraction and subsidence rate (Fig.37).

The rate of convergence of the back, and the creep of the pillars and floor will depend on numerous factors. The rheology of the various salt strata, and altered stress levels introduced due to the opening will have an effect on subsidence rate, as will the response of the overlying strata to transmit the subsidence to surface.

The representation of subsidence by the above theory requires knowledge of three site functions:

Function 1) The weighting of the contribution of the extraction within zones surrounding the point, towards the subsidence of the point.

Function 2) The function relating the influence of extraction to the long term subsidence of a point.

Function 3) The function relating the influence of extraction to initial subsidence of a point.

These functions must be determined from observation of existing subsidence survey data.

Much time and effort was spent trying to solve for these values directly by setting up and solving a series of simultaneous equations, based on data received from the CCP mine. The results proved to be voluminous and confusing. The deviation in the values derived for the variables proved to be enormous, placing no confidence in the results.

A much simpler iterative approach was arrived at to solve for the variables.

5.4. COMPUTER PROGRAM.

A computer program was written to predict subsidence based on extraction and the associated subsidence functions. A simple flowchart of the workings of the program is shown in Table #7. The required input includes the geometry of the mine workings, location of the survey stations or location of desired subsidence points, and knowledge of the three functions mentioned in 5.3.

Initially the mine plans must be modified to a form that the computer can manipulate. It was decided to represent mine plans by a block model, in which the plans are reduced to the form of a grid of blocks 50' square (Fig.38). Each block in the grid coinciding with an excavated area is coded by a mining code and the year of excavation. The mining code refers to the particular method of mining of the room, which will be assigned an extraction ratio in subsequent analysis.

A file containing the coordinates of the subsidence survey bench marks, is used to generate the circular zones of influence surrounding the point. These zones are superimposed on the block model and the proportion of mining in each zone, by year, is determined (Fig.39). The

cumulative mined proportion of each ring, by year is determined, and the weighting function associated with each ring applied. The sum of the contribution of all of the rings for a year produces an extraction factor. The functions relating extraction to subsidence rate and initial subsidence are applied to the extraction factor to determine yearly subsidence.

Output from the program consists of a data file containing initial input and numerical output, and a file containing plotting information that is used to produce graphical output. Measured subsidence may be input to compare with predicted output in numerical and graphical form.

5.5. SUBSIDENCE FUNCTIONS INVESTIGATED

5.5.1. CCP

The CCP data was initially used to derive the various site functions, necessary for input to the subsidence prediction program.

The mine plans were simplified by dividing the workings into a series of polygons, each identified by year of mining and mining type (Fig.40). All entry drifts and most of the mining panels were identified as a mining

type 1, and assigned a weight of 1.0. Some panels located in Block II employed wing rooms cut at spacings of 2-2.5 times the average room spacing. These panels were identified as mining type 2 and assigned a weight of 0.4. One panel located within the block had wing rooms cut at a spacing of 5 times the average spacing. This was identified as mining type 3 and assigned a weight of 0.2. The extraction ratio of a mining panel identified as mining type 1, assigned a weighting of 1.0, is approximately 60%. An assumption was made that a workings influence on subsidence would vary according to extraction ratio, therefore a panel of 30% extraction would be given a weighting of 0.5.

A comparison of the extraction lying within a critical radius surrounding each of 8 stations located above Block II was made with the subsidence rate experienced by each station (from Fig. 35.). All eight stations were located outside the zone of influence of active mining. The extraction considered was the 'panel' extraction and included both the mining rooms and wing pillars. Extraction was arrived at by summing the area of each mining type and multiplying by the associated weighting. The total weighted area was divided by the area of the zone of influence to arrive at an extraction. A plot of the results, shown in Figure #37, indicates a linear relationship between extraction and subsidence rate. Linear regression analysis of the plot yields the following relationship

for Function 2:

$$R = .5698X - .0523$$

where: R = subsidence rate (ft/yr)

X = extraction (mining rooms + wing pillars)

In the above analysis it was assumed that the influence of mining on subsidence was not dependent on the distance of the excavation from the subsiding point. Contribution towards subsidence was identical for any unit excavation located within the zone of influence. Function 1, the zone weighting, was defined such that every extracted point lying within the zone of influence had equal weight.

Setting the unknown Function 3 to zero the values were introduced into the subsidence program and plots of predicted and actual subsidence were generated, as shown in Fig 41. The initial years actual and predicted subsidences don't correspond very closely, however comparing the later years results we see that the difference between successive years subsidence is fairly close for predicted and actual.

The effect of the introduction of a value for Function 3 is shown in Figure 42. The shape of the function was assumed to be linear as follows:

$$S = 1.50X$$

where S = subsidence attributed to function 3

X = extraction (mining rooms and wing pillars)

The value of the function was adjusted until a best fit was achieved visually. The effect of considering time lags of 0.5, and 1.0 years between initial mining and the full contribution of the effect of Function

3 on subsidence are shown in Figure's 43 and 44. A decrease in the lag time has the effect of shifting the predicted subsidence trough in the direction of mining.

A comparison of the predicted and actual curves for the case of the CCP data shows fair correlation in general shape and magnitude of subsidence at the east and west ends of the line. The centre portion however is constantly underpredicted. There are several explanations that could account for this discrepancy :

1)In the 1978 survey a closure error of approximately 0.2 feet was made. The error was distributed among the survey stations. If we discount the 1978 survey for reasons of questionable accuracy, and consider only data from 1979 and later, predicted and observed subsidences show remarkable correlation (Fig.45). The shape of the predicted subsidence as well as the magnitudes for the various years are quite close to the observed values.

2)Some of the mining panels in the west end of Block II had to be terminated due to the encounter of brine inflow. The response of the rooms, pillars, and overlying salt strata in these brine areas should be different than those rooms not experiencing brine inflow. It is assumed for the analysis that the mining horizon is of a homogeneous material, in which all extracted areas behave identically. No consideration was made to weight the contribution of these areas of brine inflow differently from the rest of the workings.

3)Differences in salt rock properties and composition undoubtedly exist within the mining level. The rheologies of these different compositions will also differ. No account of the differences in the rock composition or their different responses to mining are taken into account.

5.5.2. LANIGAN

When dealing with the Lanigan data, it was decided to weight the excavation relative to a base extraction of 60%, for comparison with the CCP results. For example a room and pillar working of 34% extraction was assigned a weight of 34/60=.583.

As there were no stations undermined that were lying outside the area of influence of the active mining, the function relating extraction to subsidence rate couldn't be determined directly. An iterative approach was used to determine the various functions, using the factors derived from the CCP case as a base. Direct substitution of the CCP parameters result in the predicted curves indicated in Figure 46.

Using the CCP parameters a poor fit of predicted to actual subsidence results. The predicted subsidence trough is much shallower than the actual observation, and has steeper sides. This indicates the outer ring zone weightings too large, and inner rings weightings are too small.

The shape of the influence function used with the CCP data assigns equal weight of influence to all elements of extraction surrounding a point and is shown graphically in Figure 47. The shape of some of the influence functions listed in table 5 are also plotted.

The effect of weighting the zones according to equation 6, an asymptotic influence function derived from observational and probability considerations, and equation 3, an asymptoic influence function based on probability considerations, are shown in Figure 48 and 49. Equation 3, which is more heavily weighted close to the subsiding point, produces a steep subsidence trough, while equation 6 produces a trough closely resembling the observed results. Adjusting function 2 to a value of 0.4X and function 3 to a value of 0.2X with a 1/2 year lag, produces a fairly close correlation between predicted and observed results as shown in Figure 50.

There is some difference between the predicted and observed subsidence at the SW end of the surveyed line. This could be attributed to the accuracy of the surveys, location of the reference bench mark, or limitations of the program. The 1981 subsidence survey indicated uplift was occuring at a point some 7000 feet SE of the workings, while the reference bench mark was located only 4000 feet from the edge of the workings. In 1983 subsidence was recorded at the same point located 7000 feet from the workings. This would indicate that the reference bench mark was actually located in an active area, and was probably experiencing uplift during those years.

It could be expected that upon excavation of the mining zone, the beam action of the overlying sedimentary rock could cause uplift of points surrounding the workings. Depending on the strength of this overlying material and the flexing of the strata, this uplift might eventually subside as the strata fractures and settles under load.

5.5.3. CORY

There was an insufficient history of subsidence and mining data available to directly solve for the various functions, so an iterative approach was used to derive the relationships.

It must be noted that in the 1982 and 1983 surveys the two stations located at the bottom of the subsidence trough were destroyed. Straight lines connect all existing survey stations on subsequent plots.

Direct substitution of the CCP functions resulted in the fit as shown in Figure 51. The predicted subsidence trough was shallow compared to the observed subsidence, and subsidence was overpredicted at the east and west ends of the survey line. This indicates the weighting of the outer rings is too large for this situation. Employing the influence function used with the Lanigan data results in a much closer fit as shown in Figure 52. The effect of setting the lag time to 0 and 1 year is shown in Figure 53 and 54.

It should be noted that the 1982 and 1983 surveys indicate subsidence occuring at a distance of 10000 feet to the west of the workings. These points are located well away from the range of the influence of the workings. It is suggested that reference bench mark 2B, located at the shaft has actually undegone some uplift.

5.6. DISCUSSION

The influence function defined by equation 6 of Table 5, was found to adequately represent the shape of the subsidence curves of the Cory and Lanigan mines. When this influence function is applied to the case of the CCP data, it is found to correlate fairly well (Fig.55). The shape of the CCP subsidence trough seems much less sensitive to weighting of the influence function than those of Cory or Lanigan. The overburden geology and mining depths of the workings are fairly similar so it can be expected that the propagation of subsidence from the workings to surface would occur in a similar manner for the three mines.

Similar functions are used to describe the relationship of extraction to subsidence for the Cory and CCP mines which could be expected as they both employ similar mining methods and extractions.

The functions relating subsidence to extraction for the Lanigan mine, are different from those of the Cory and CCP lines, which could be attributed to the different mining method.

There is a much larger initial contribution to subsidence caused by chevron panel mining than by the pillar panel mining. A possible explanation is that the chevron mining technique is designed to allow the wing rooms to collapse, thereby relieving stress in the area of the advancing entries. This technique is expected to allow a much larger volume of rock to move into the openings than in the pillar panel method. On the other hand the roof of a pillar panel mining room may close in a much more controlled fashion. The narrower pillars separating the rooms will deform more uniformly than the wide pillars separating the chevron panels, allowing the back to converge in a uniform manner.

Most of the room heights of the workings range between 10.5 and 12 foot high. The more recent rooms at the Lanigan mine have been excavated to a 16 foot height. It is not presently known what the effect of this room height has on the subsidence as not enough data has been compiled. All the functions used are based on two dimensional extraction.

6. CONCLUSIONS AND RECOMMENDATIONS

1) The total subsidence of ground above Saskatchewan potash mines can be represented by two stages of development. At some time after the mining of an excavation an initial component of subsidence is experienced on surface, followed by a long term constant rate of subsidence.

2) The magnitude of the initial effect of subsidence and the rate of long term subsidence, are functions of the mining extraction and mining type.

3) Mines employing the same method of mining experience similar types of subsidence.

4) The zone area method of subsidence prediction has reproduced actual subsidence to a fair degree of accuracy for the limited number of available test cases.

5) Use of the zone area method of subsidence prediction requires that the site functions relating initial and long term subsidence to extraction are known, along with the influence function relating effect of subsidence with distance.

6) Recommended subsidence survey procedures are listed in sections 4.2. and 4.4..

The conclusions listed are based on experience with a limited number of test cases, and sufficient data is not presently available from other sources for comparison. In the future, additional data will become available that may be able to test the accuracy of the present observations.

In this analysis certain areas have not been considered, and future studies should address the following:

- the effect of room/pillar height
- the effect of sequencing of mining
- the effect of different rock types in the mining zone and overburden.
- the location and effect of discontinuities in the mining zone and overburden.
- the effect of earth quakes

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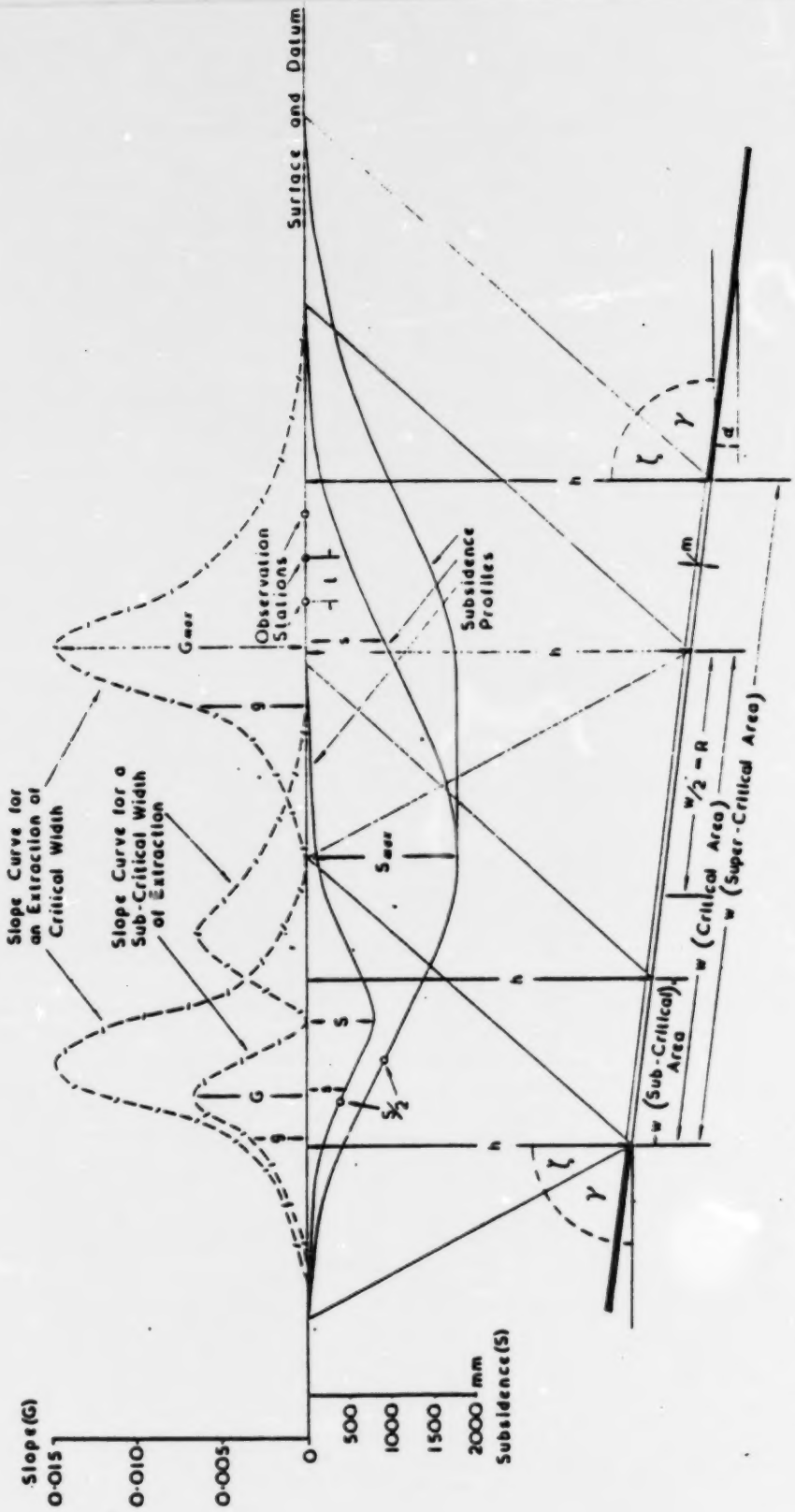


Fig. 1. Trough subsidence profile
(From N.C.B. Handbook (52))

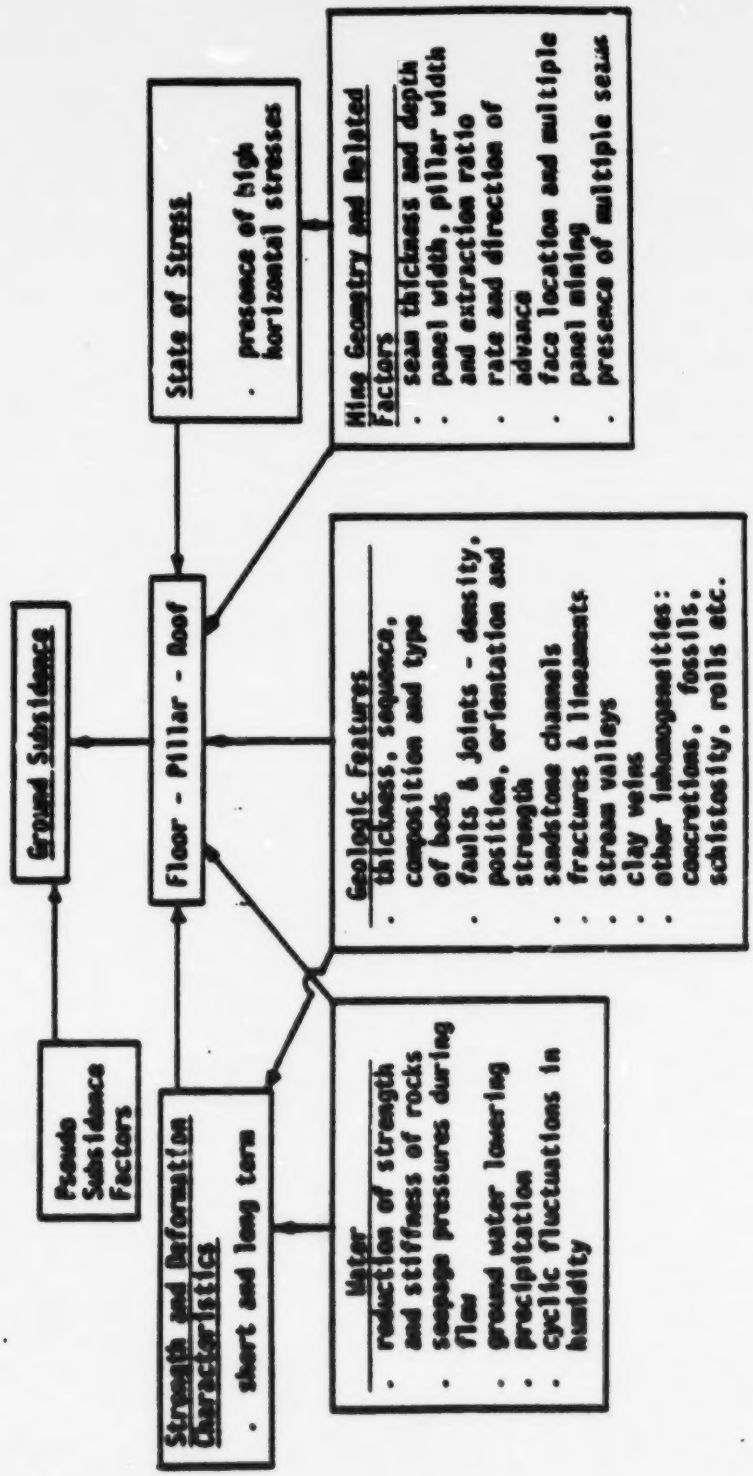


Fig. 2. Factors contributing to ground subsidence
(From Mather and Mikkilineni (46))

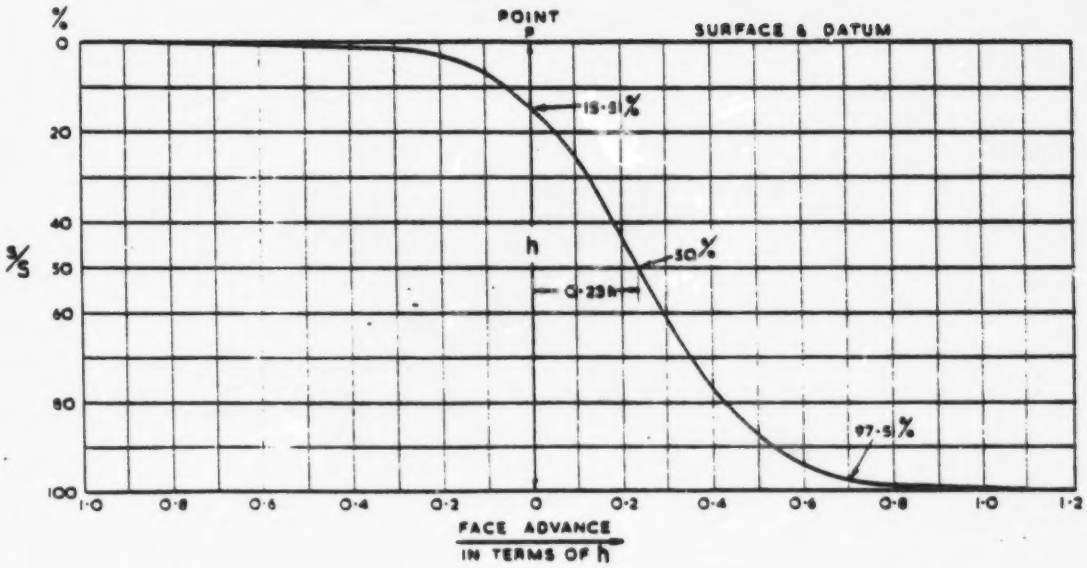


Fig. 3. Subsidence development curve
(From N.C.B. Handbook (52))

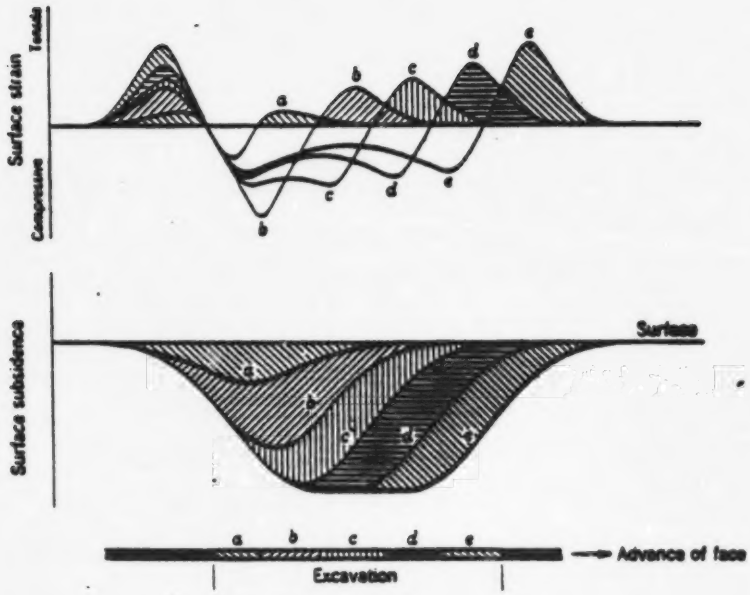


Fig. 4. Progress of subsidence with advance of excavation area (From Obert and Duvall (53))

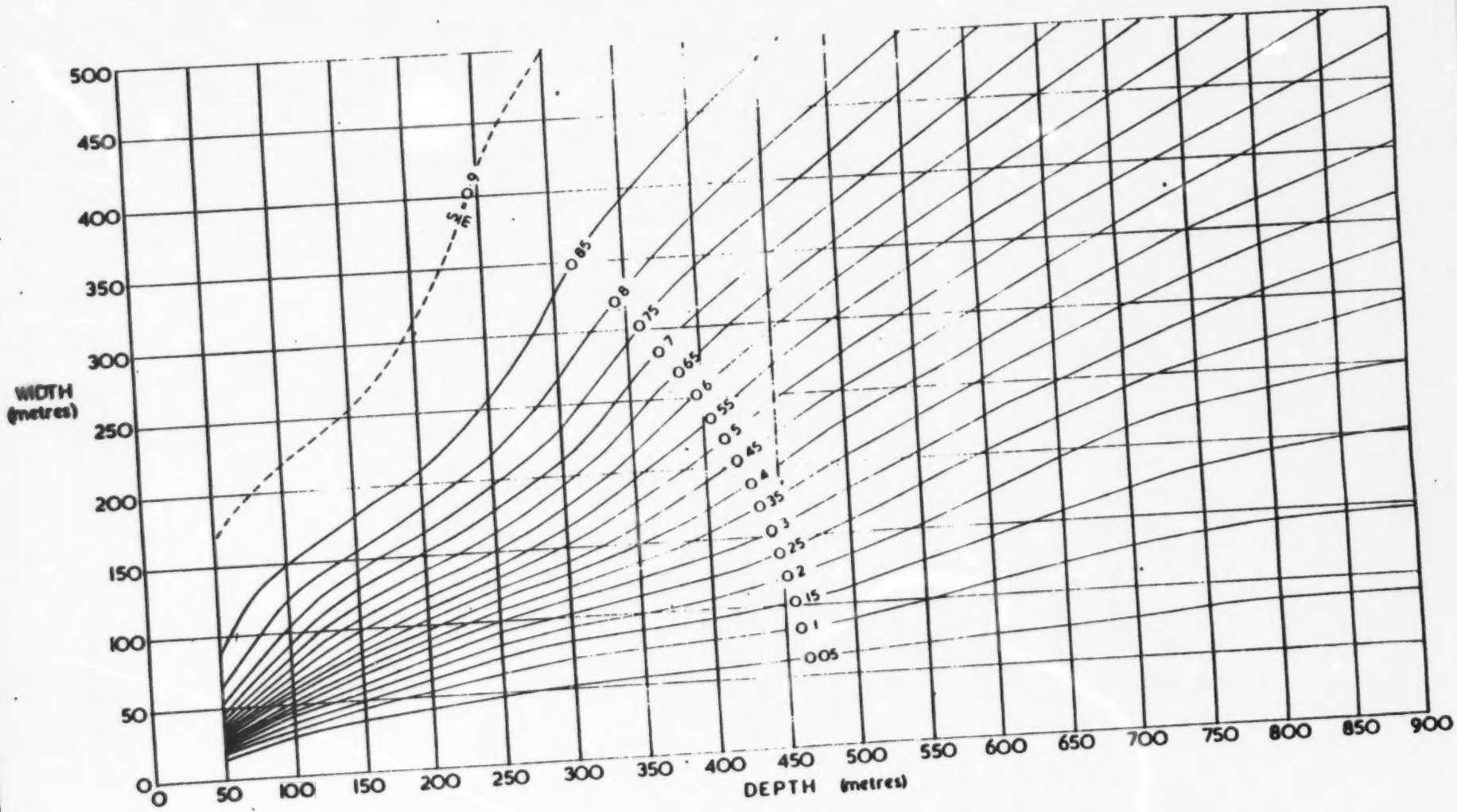


Fig. 5a. Prediction of subsidence - Maximum subsidence
(From N.C.B. Handbook (52))

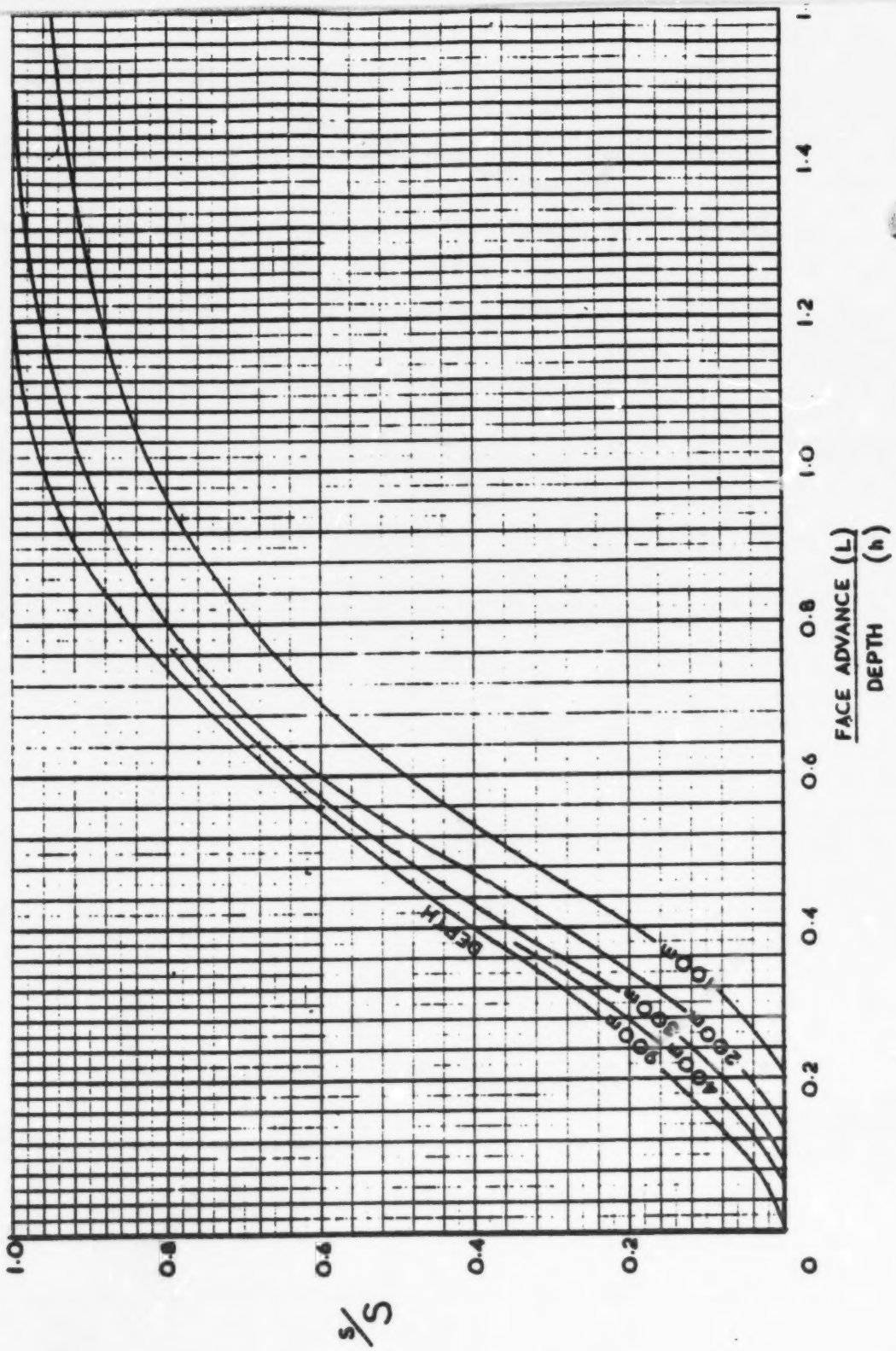


Fig. 3b. Prediction of subsidence - Correction for incomplete face advance. (From N.C.B. Handbook (52))

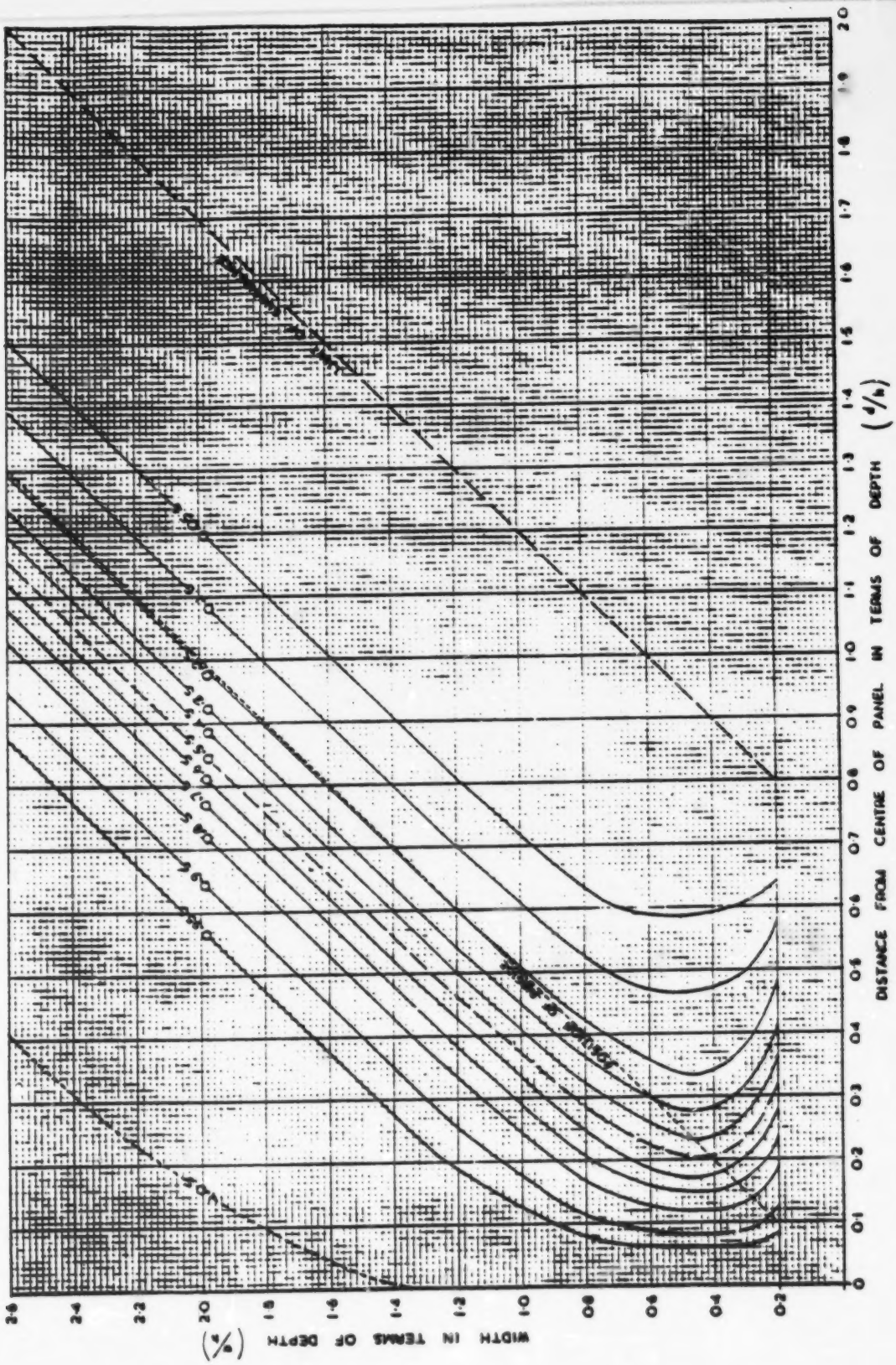


Fig. 5c. Prediction of subsidence - Subsidence profile
(From N.C.B. Handbook (52))

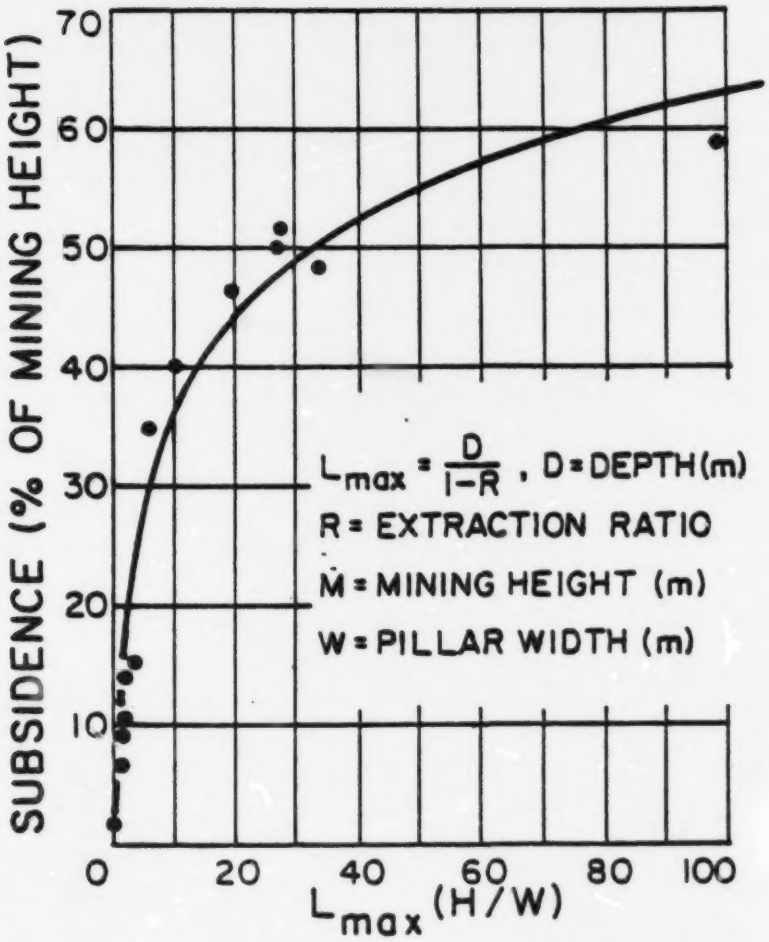


Fig. 6. Subsidence above room and pillar workings
Subsidence (% of Mining Height) VS. Lmax(D/(1-R))
(From Mather and Mikkilineni (46))

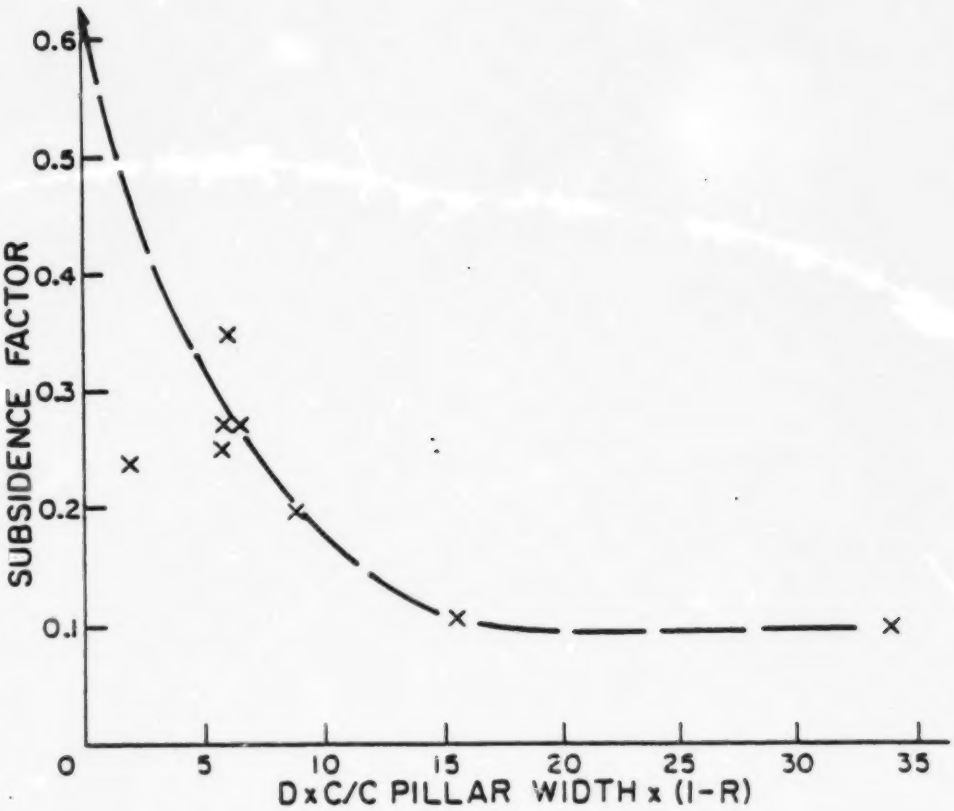


Fig. 7. Subsidence above room and pillar workings
Subsidence Factor VS. (D × C/C Pillar Width × (1-R))
(From Mather and Mikkilineni (46))

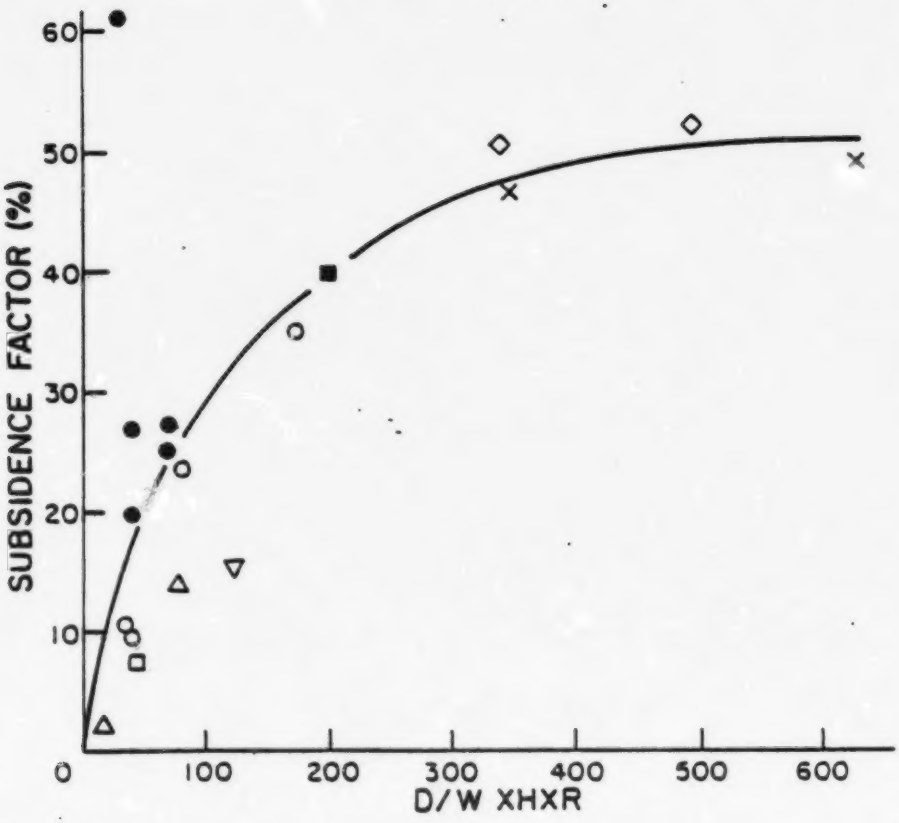


Fig. 8. Subsidence above room and pillar workings
Relationship between Subsidence Factor and (D/W)×H×R
(From Mather and Mikkilineni (46))

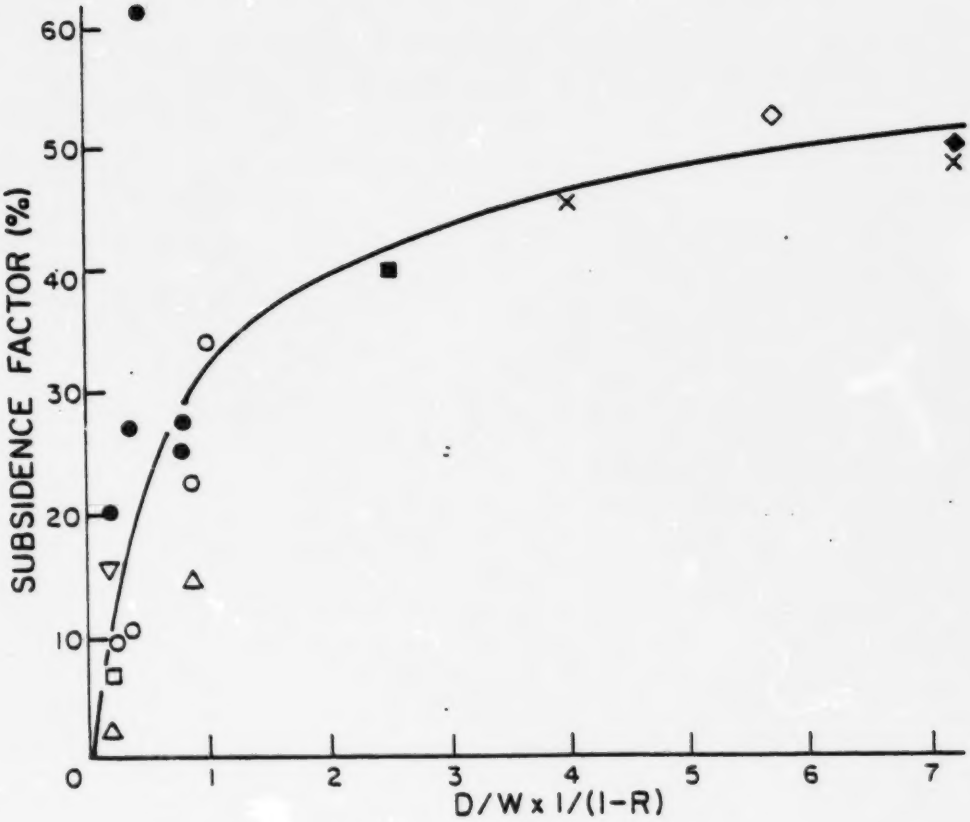


Fig. 9. Subsidence above room and pillar workings
Relationship between Subsidence Factor and D/W×1/(1-R)
(From Mather and Mikkilineni (46))

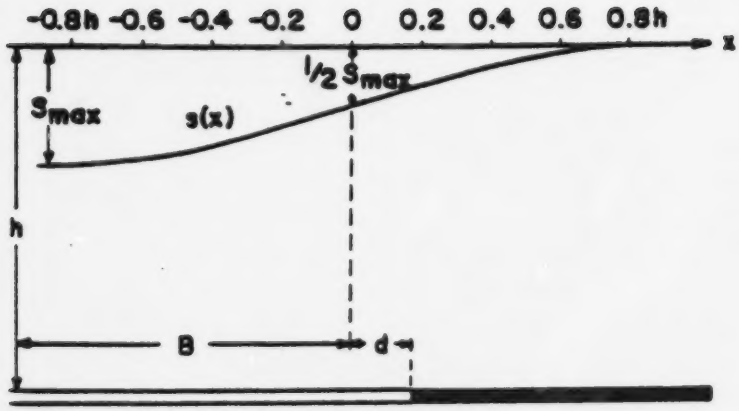


Fig. 10. Subsidence profile function
(From Braeuner (11))

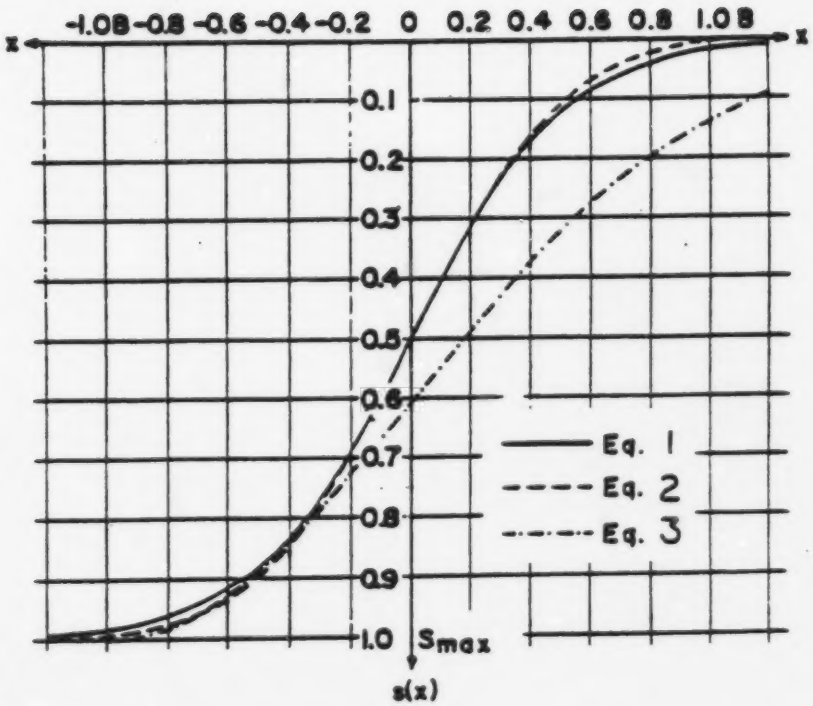


Fig. 11. Asymptotic profile function
(From Braeuner (11))

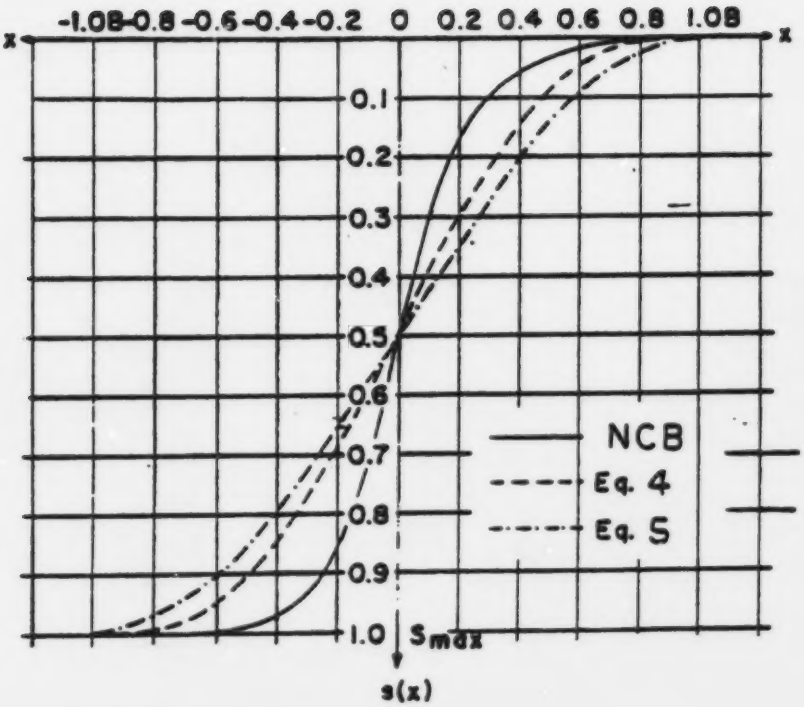


Fig. 12. Nonasymptotic profile function
(From Braeuner (11))

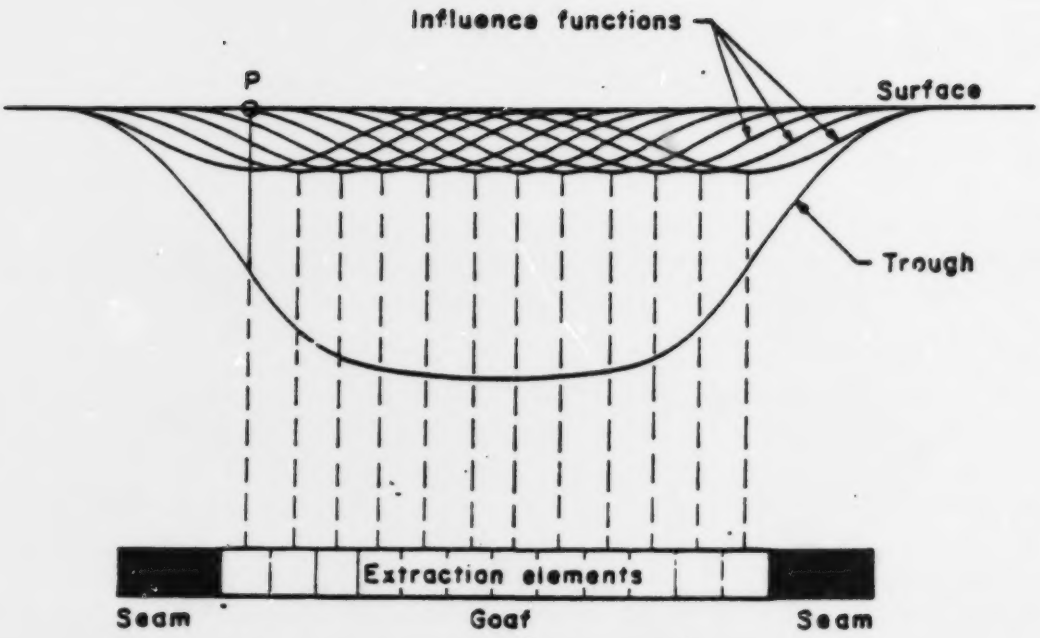


Fig. 13. Superposition of influence functions
(From Braeuner (11))

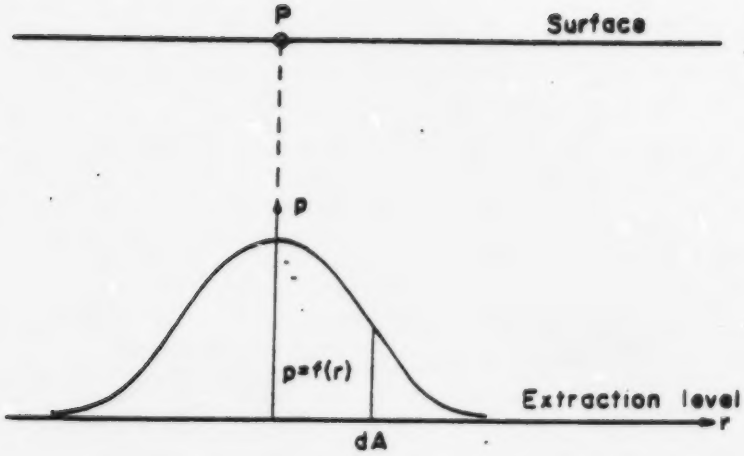


Fig. 14. Influence function
(From Braeuner (11))

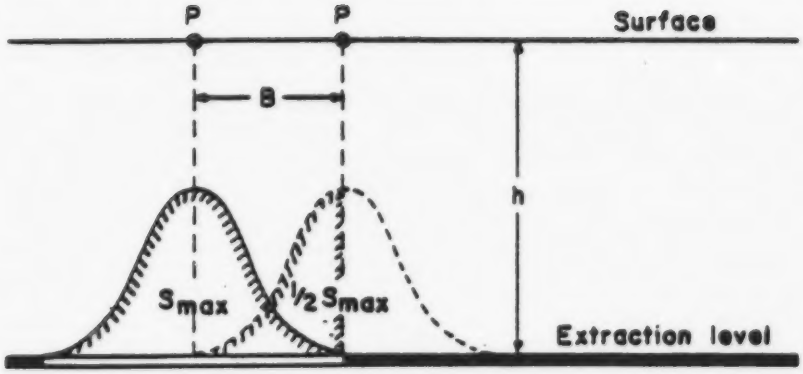


Fig. 15. Influence function and subsidence
(From Braeuner (11))

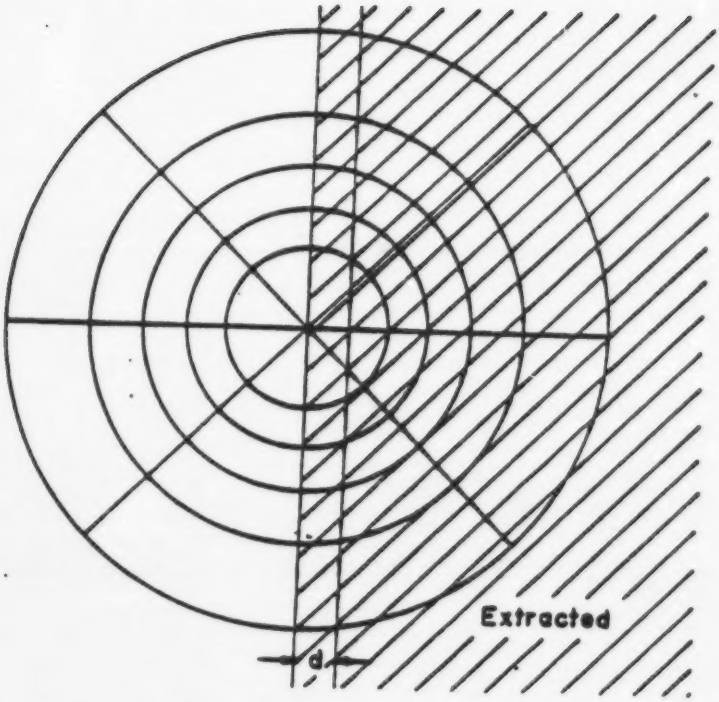
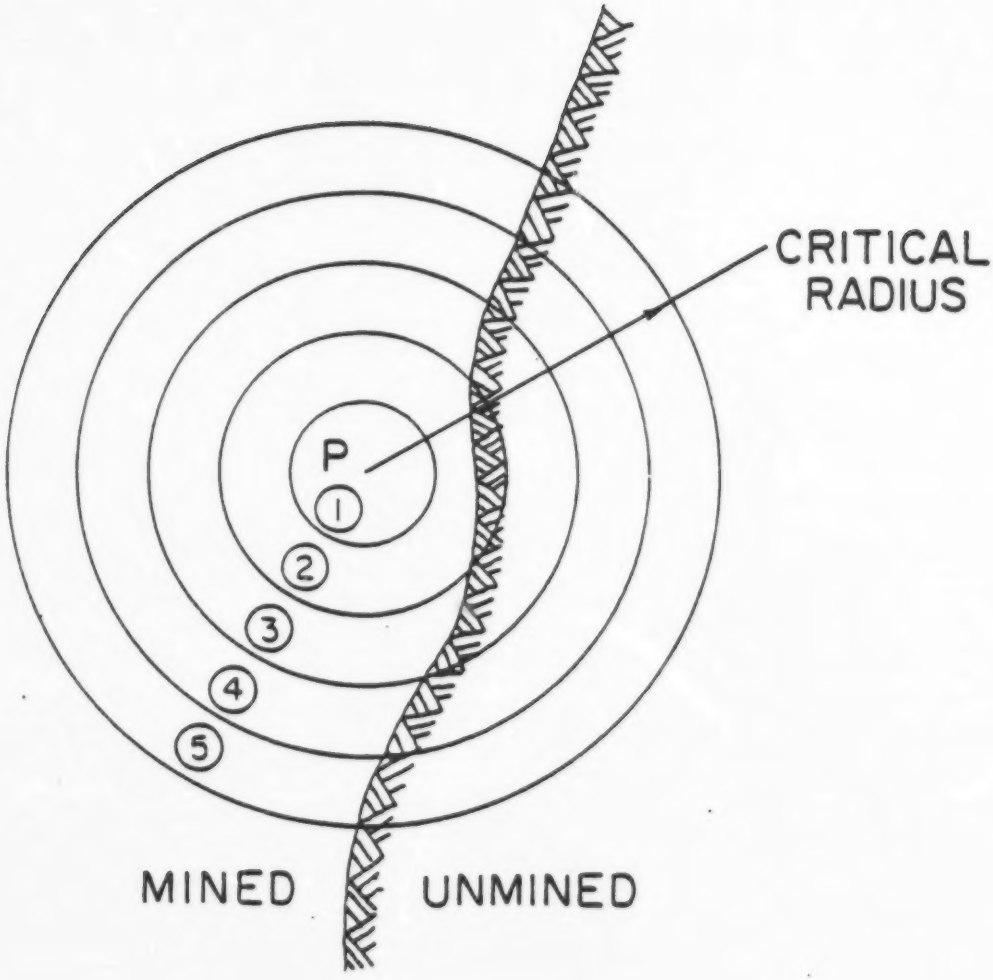


Fig. 16. Compensation zone
(From Braeuner (11))



A = proportion of zone #1 excavated
B = proportion of zone #2 excavated
C = etc.

a = influence factor of zone #1
b = influence factor of zone #2
c = etc.

Subsidence at point P is determined by:

$$S(p) = Aa + Bb + Cc + Dd + Ee$$

Fig. 17. Manual calculation of influence functions

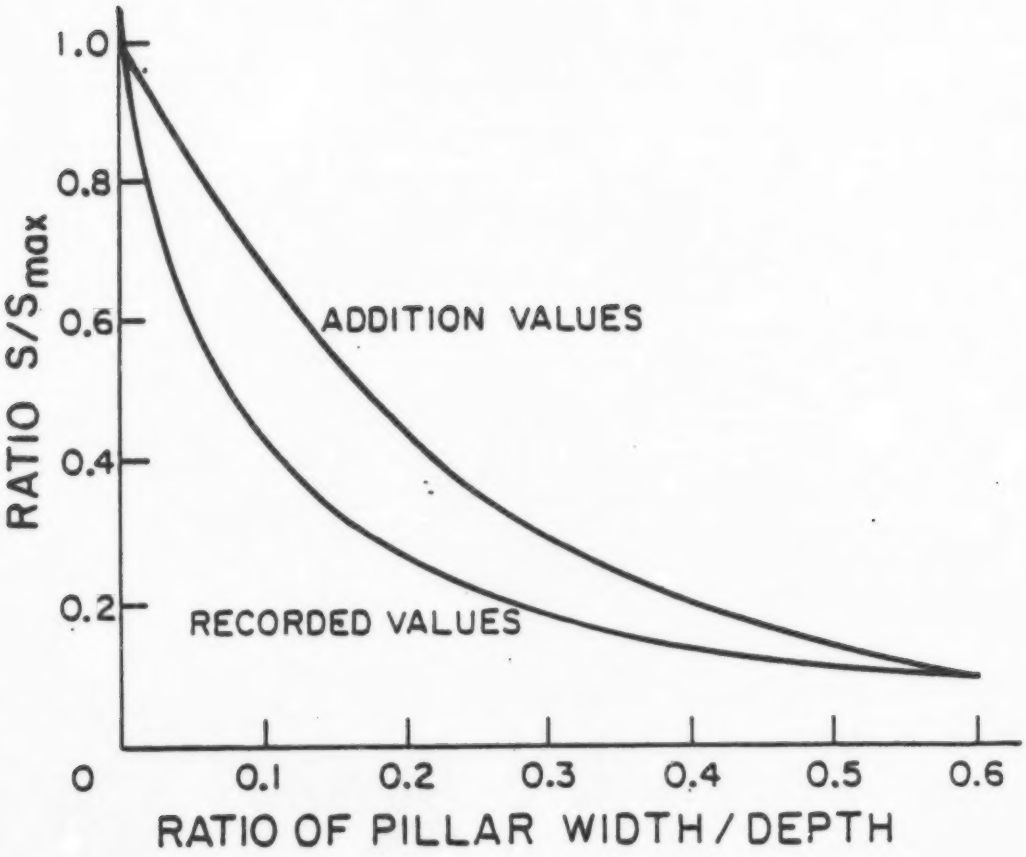


Fig. 18. Coal pillar reduction effects
(From Marr (45))

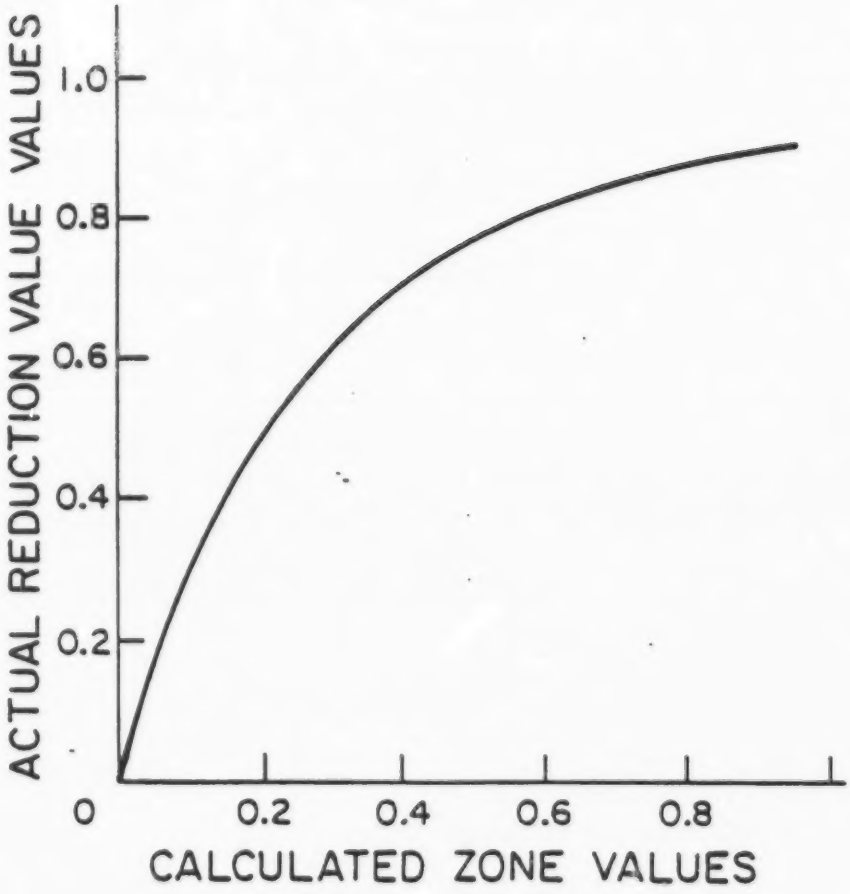


Fig. 19. Reduction Factor correction
(From Marr (45))

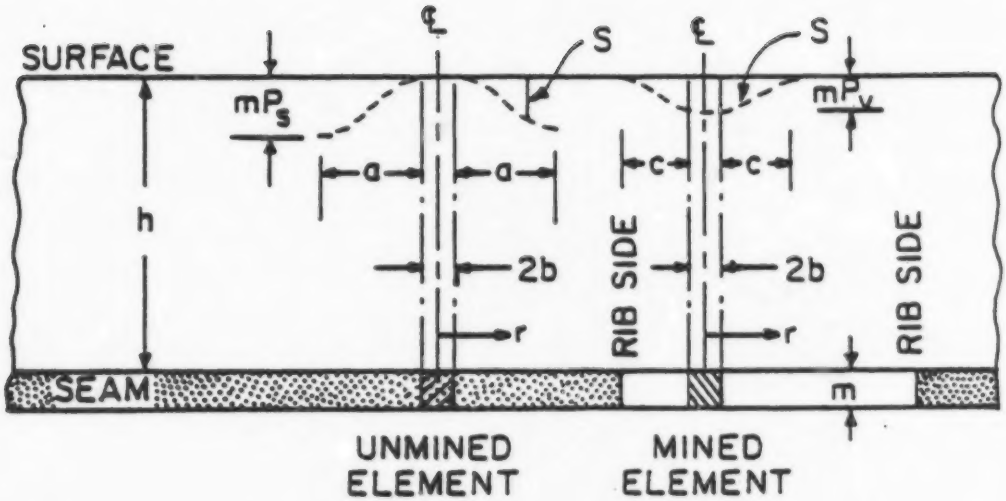


Fig. 20. Complementary elements
(From Sutherland and Munson (75))

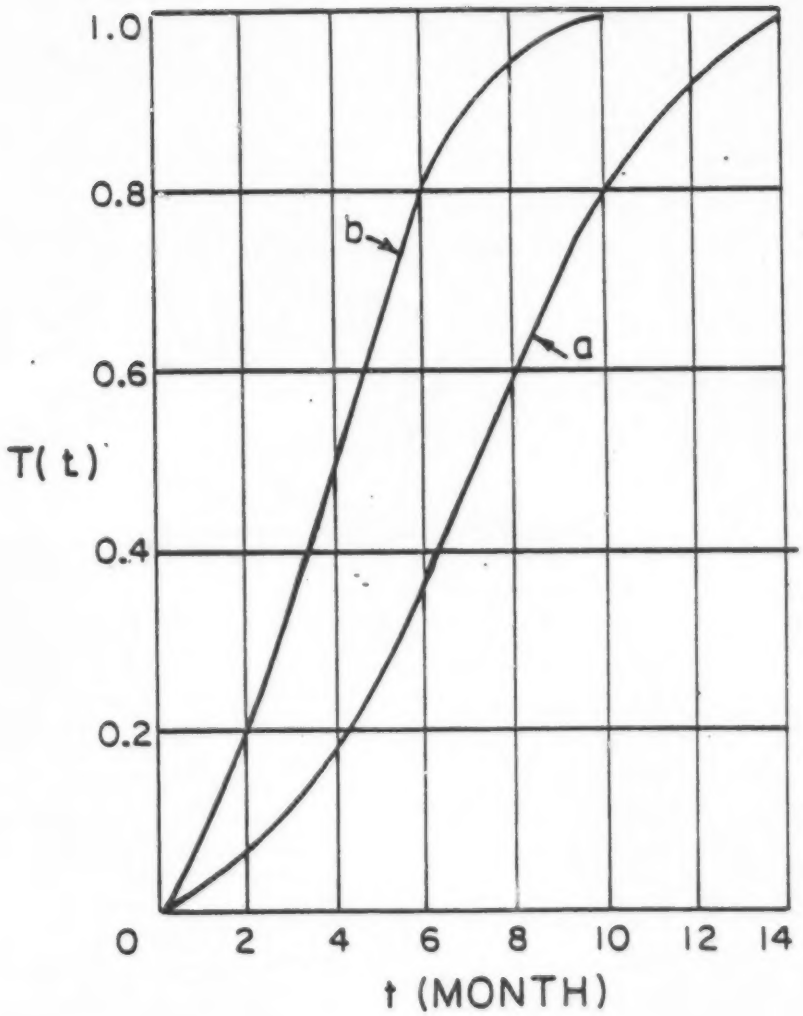


Fig. 21. Time factors
(From Hiramatsu and Oka (32))

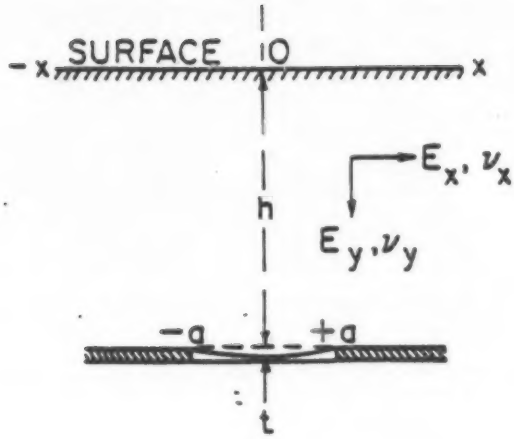


Fig. 22. Thin slit in transversely isotropic plate
(From Obert and Duvall (53))

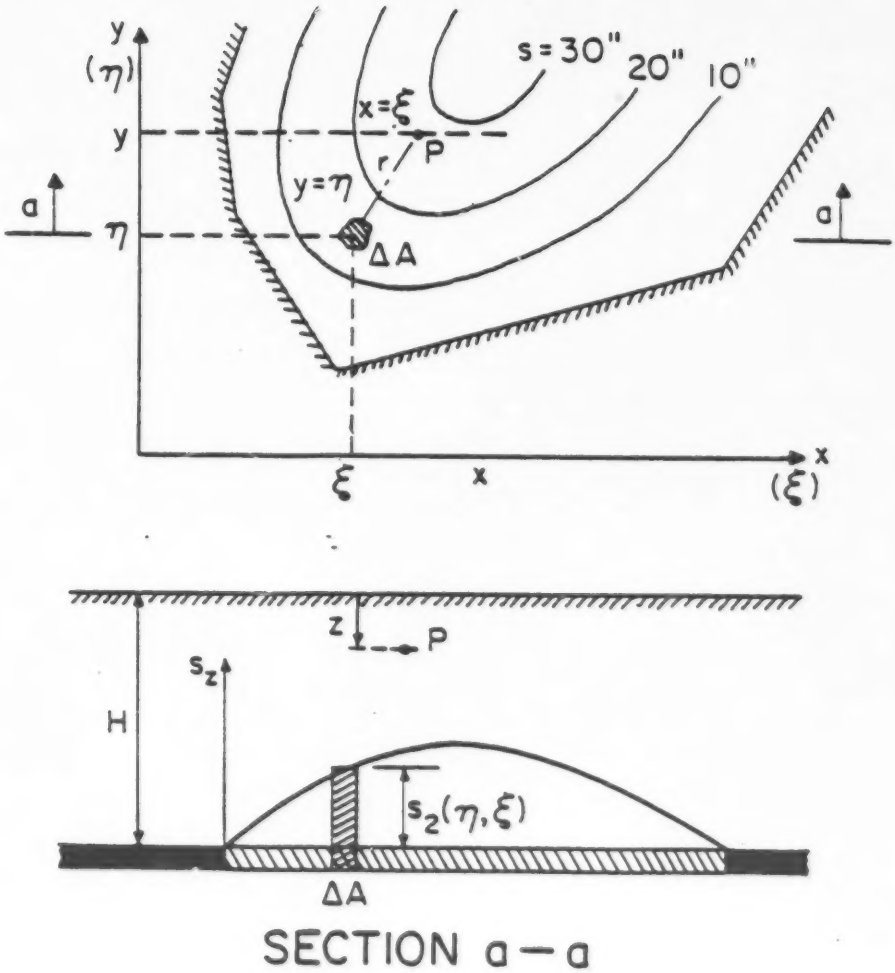
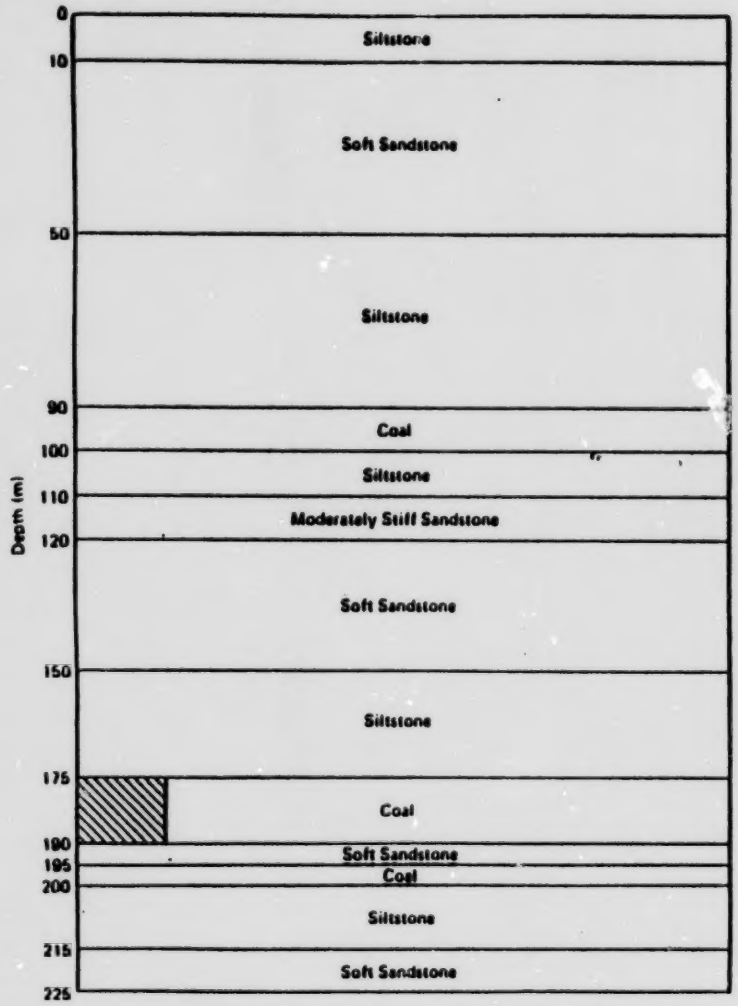
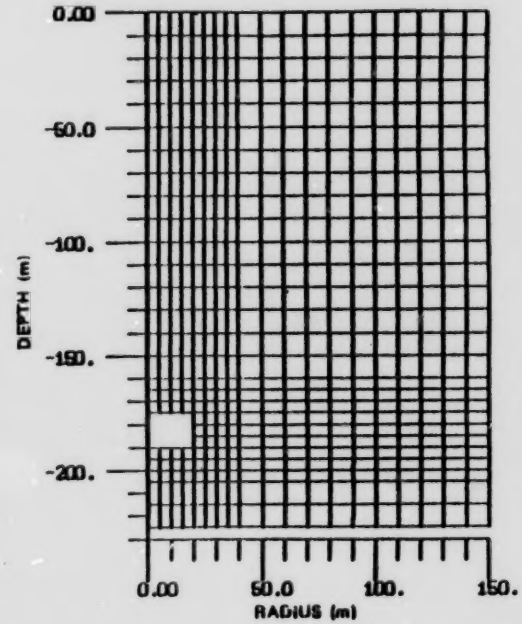


Fig. 23. Face element idealization
(From Salamon (60))



Stratigraphic column for the Centralia UCG plane-strain subsidence model.



STEALTH plane-strain calculational grid used for the Centralia subsidence studies.

Fig. 24. Stratigraphic column and its finite element mesh (From Trent (82))

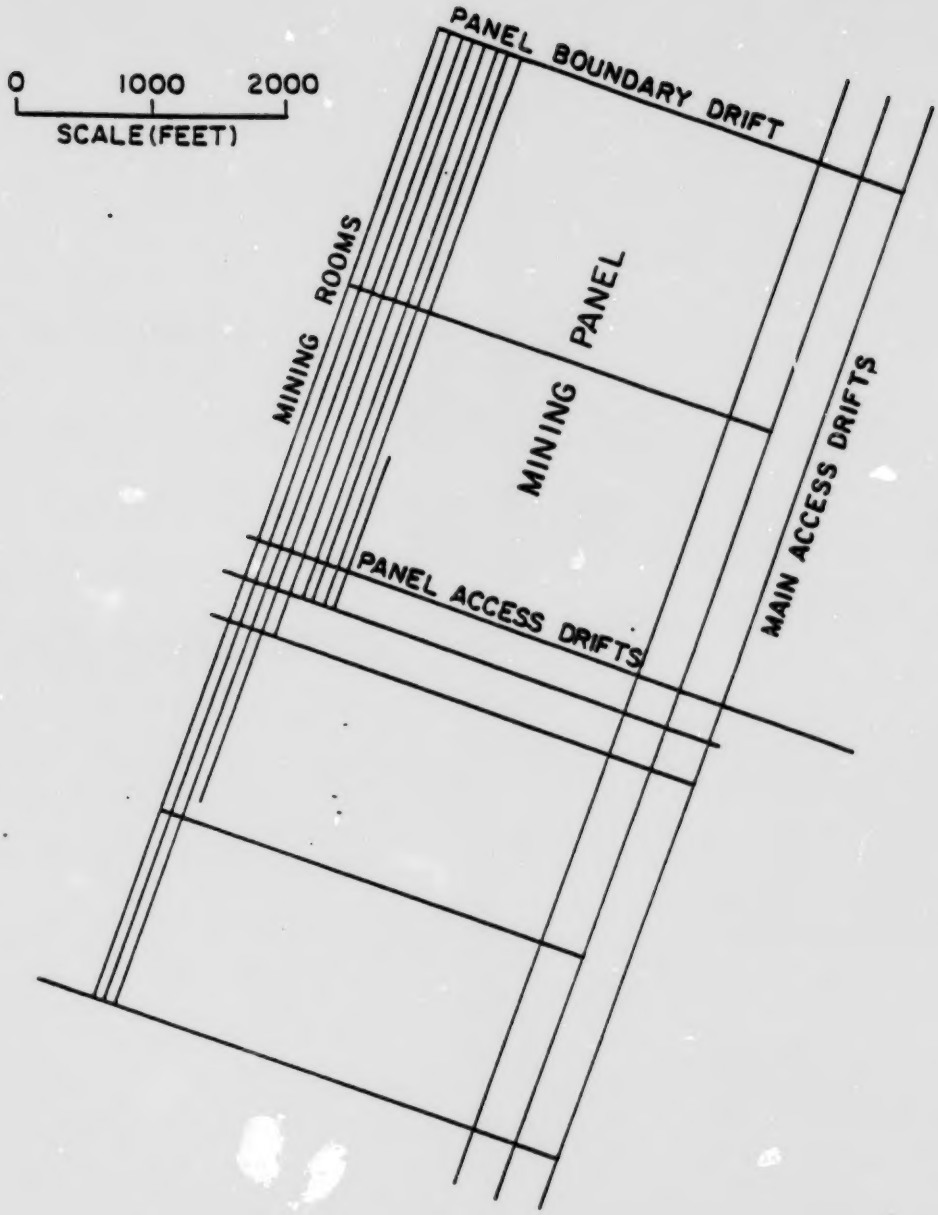


Fig. 25. Pillar panel mining layout

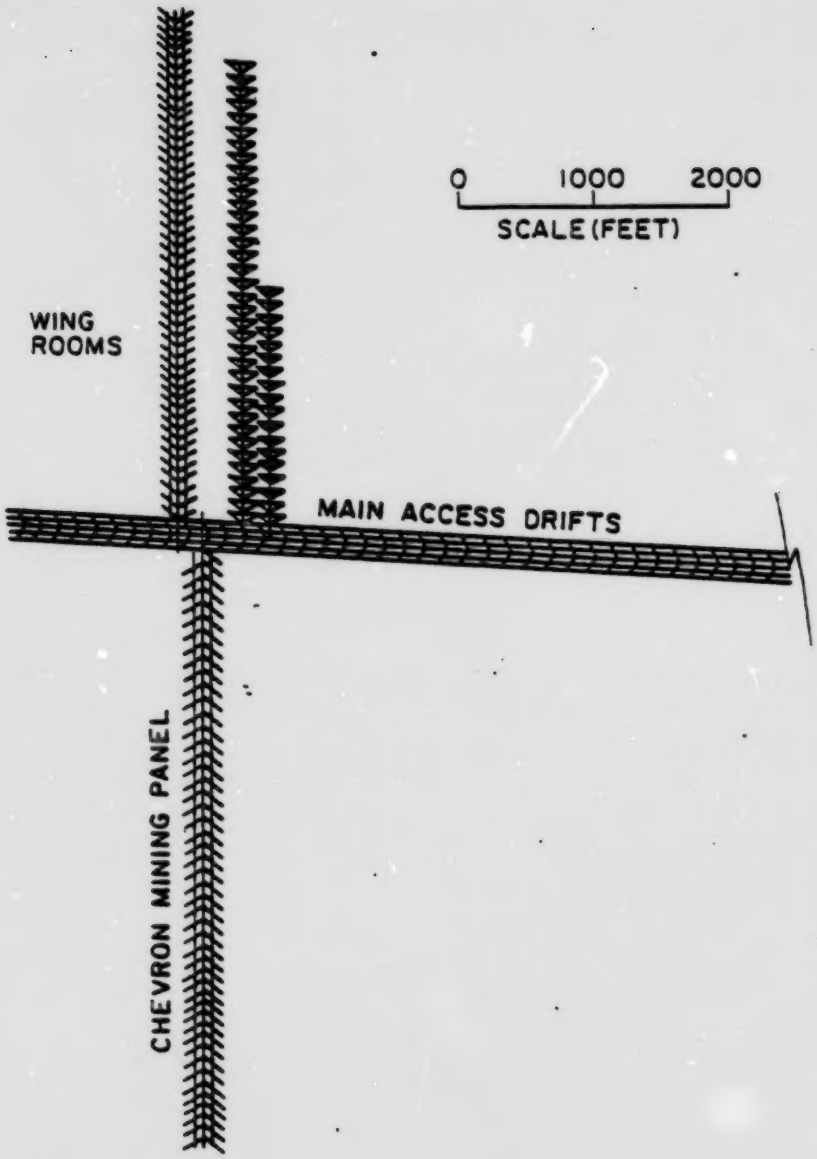


Fig. 26. Chevron mining panel

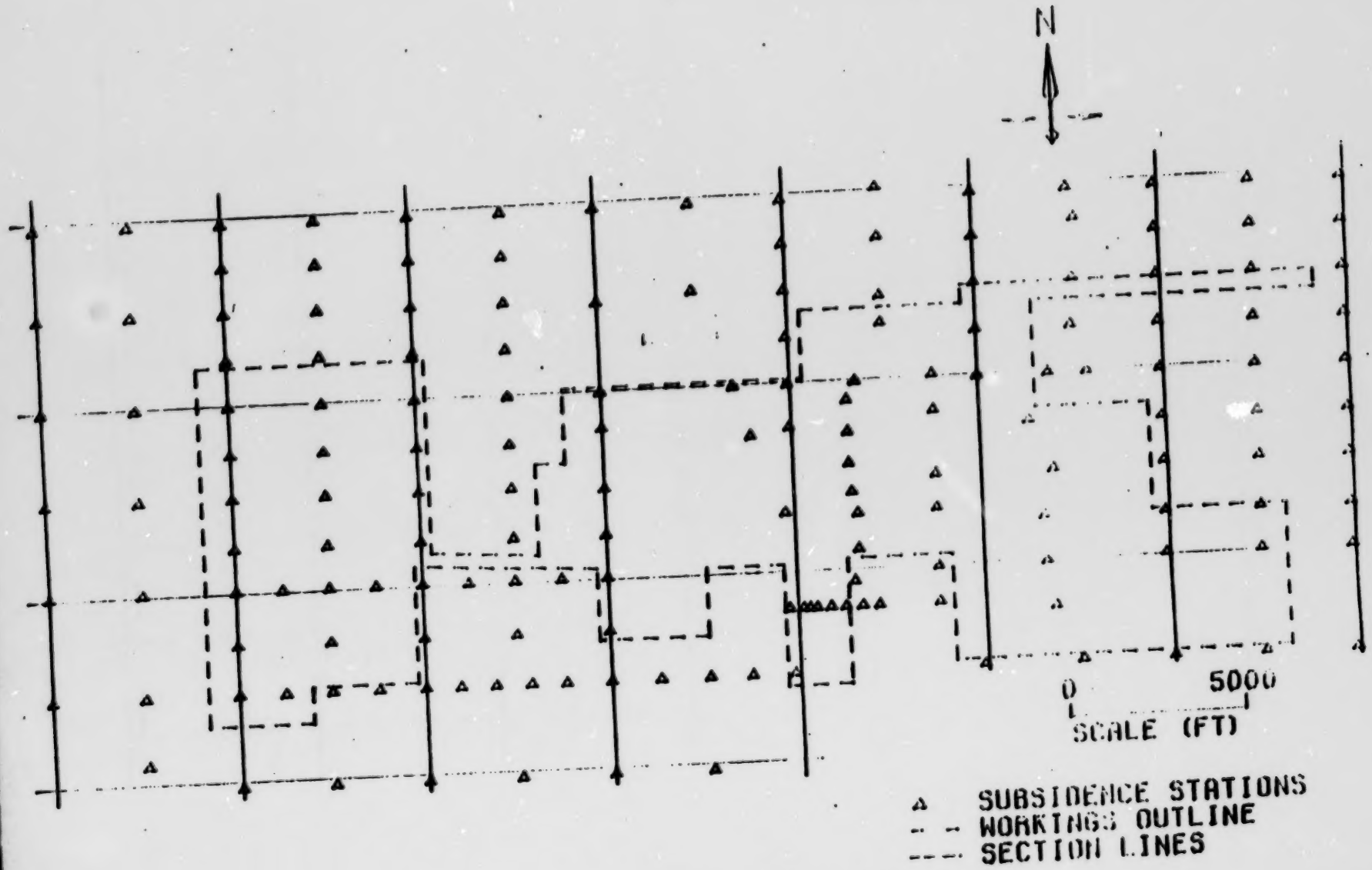
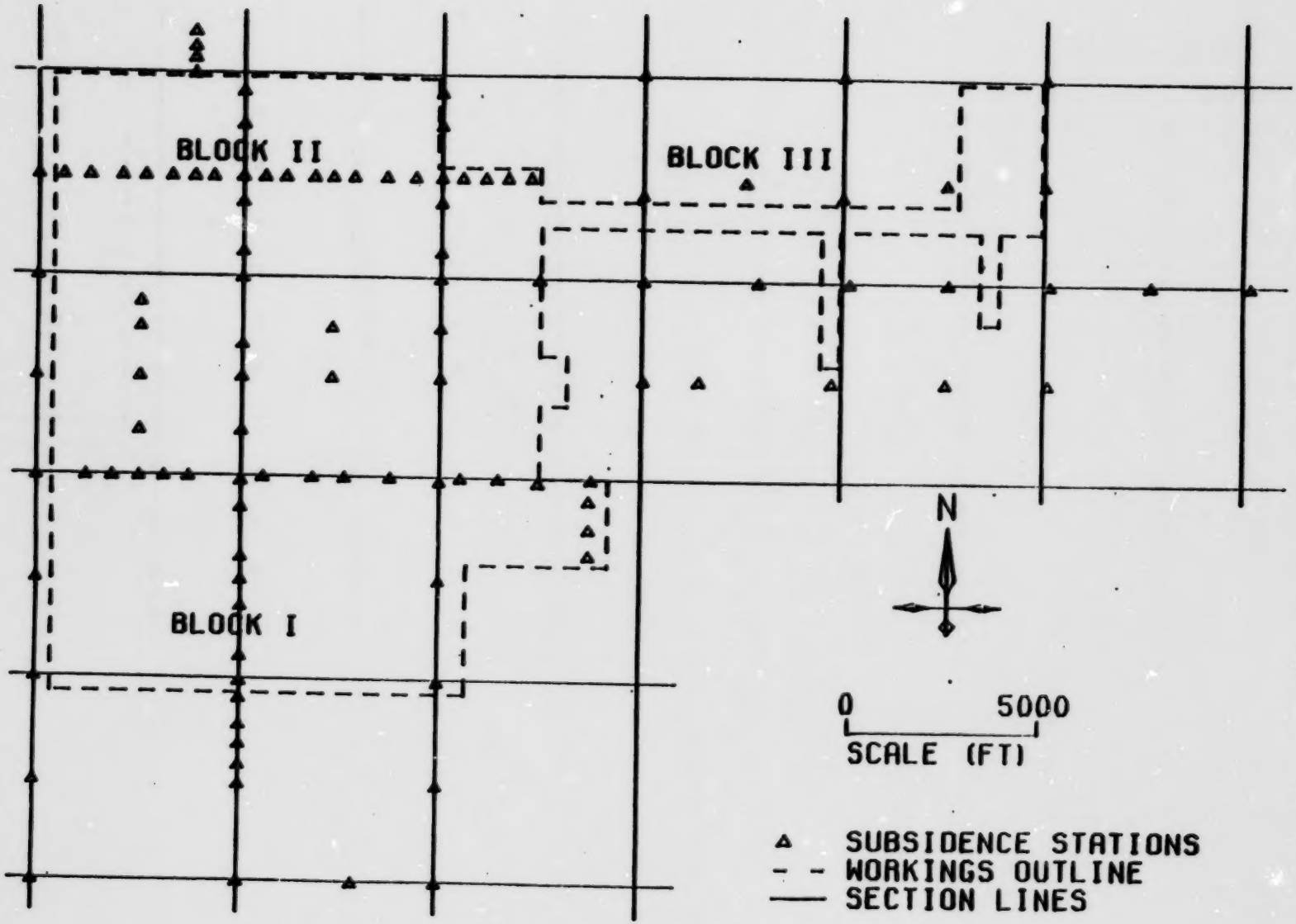


FIG.27. ALLAN SUBSIDENCE STATIONS


BLOCK II
BLOCK III
BLOCK I
N
0
5000
SCALE (FT)
SUBSIDENCE STATIONS
WORKINGS OUTLINE
SECTION LINES

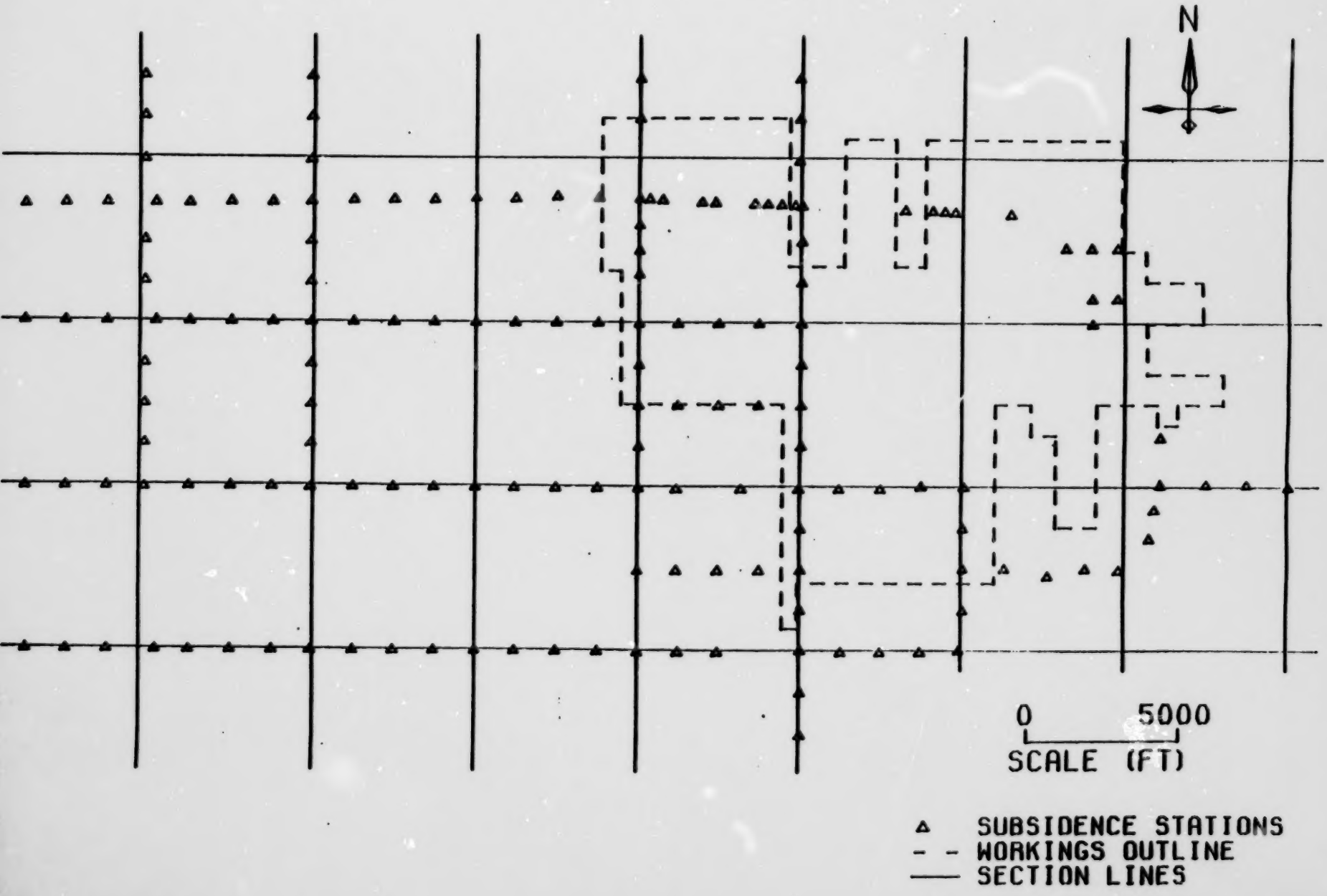


FIG 28 SODY SUBSIDENCE STATIONS

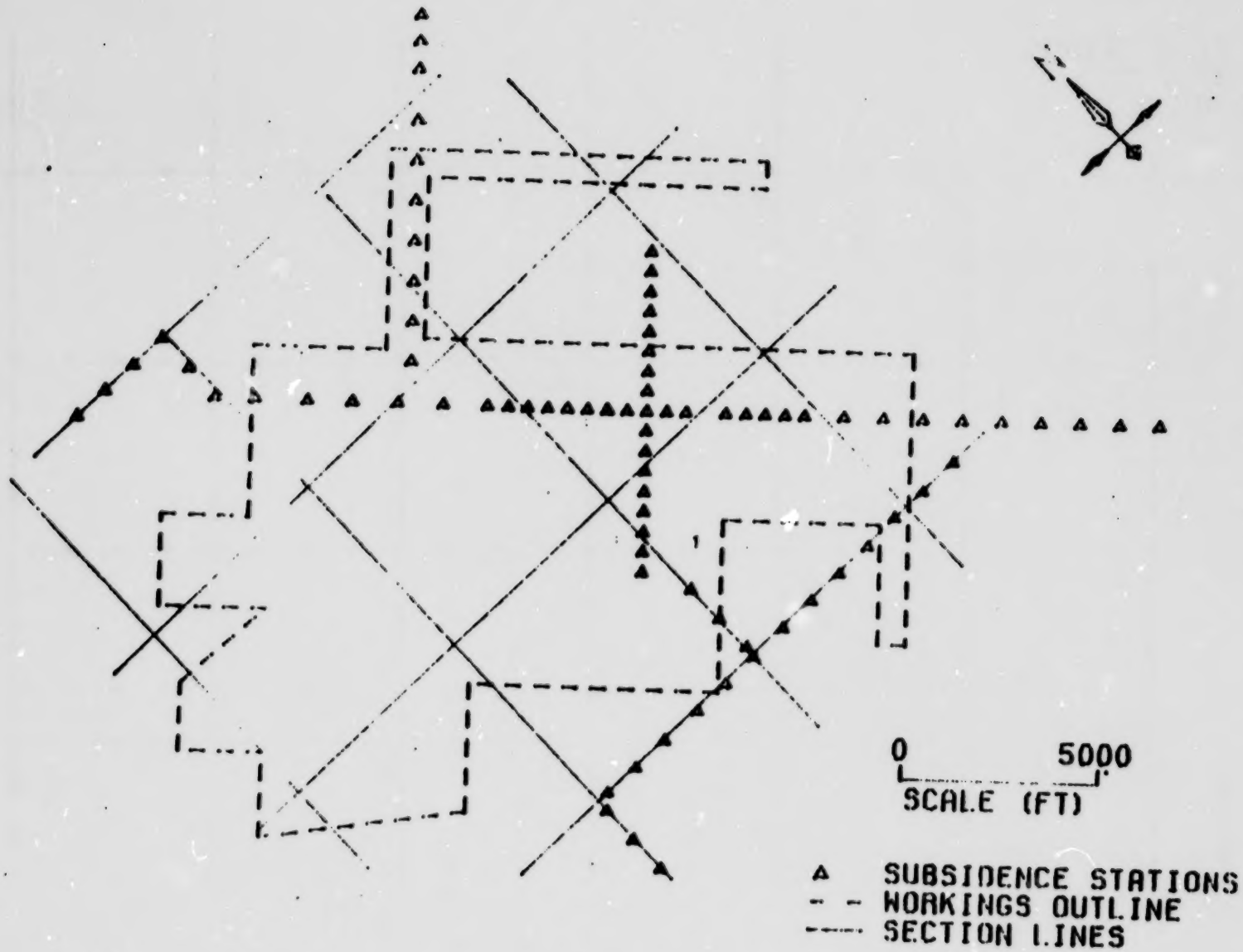
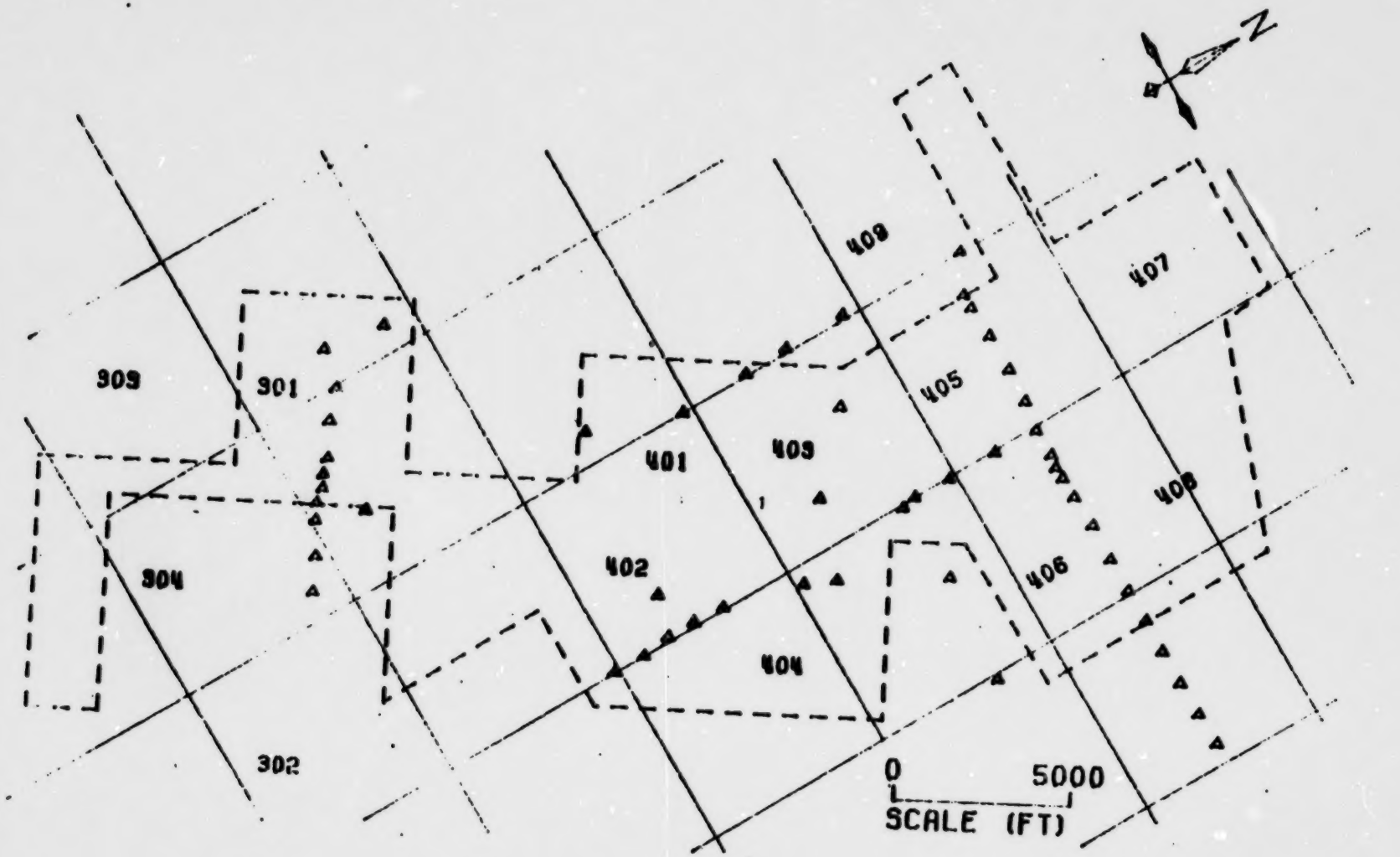


FIG.30. LANIGAN SUBSIDENCE STATIONS



△ SUBSIDENCE STATIONS
- - WORKINGS OUTLINE
—— SECTION LINES

FIG.31. ROCANVILLE SUBSIDENCE STATIONS

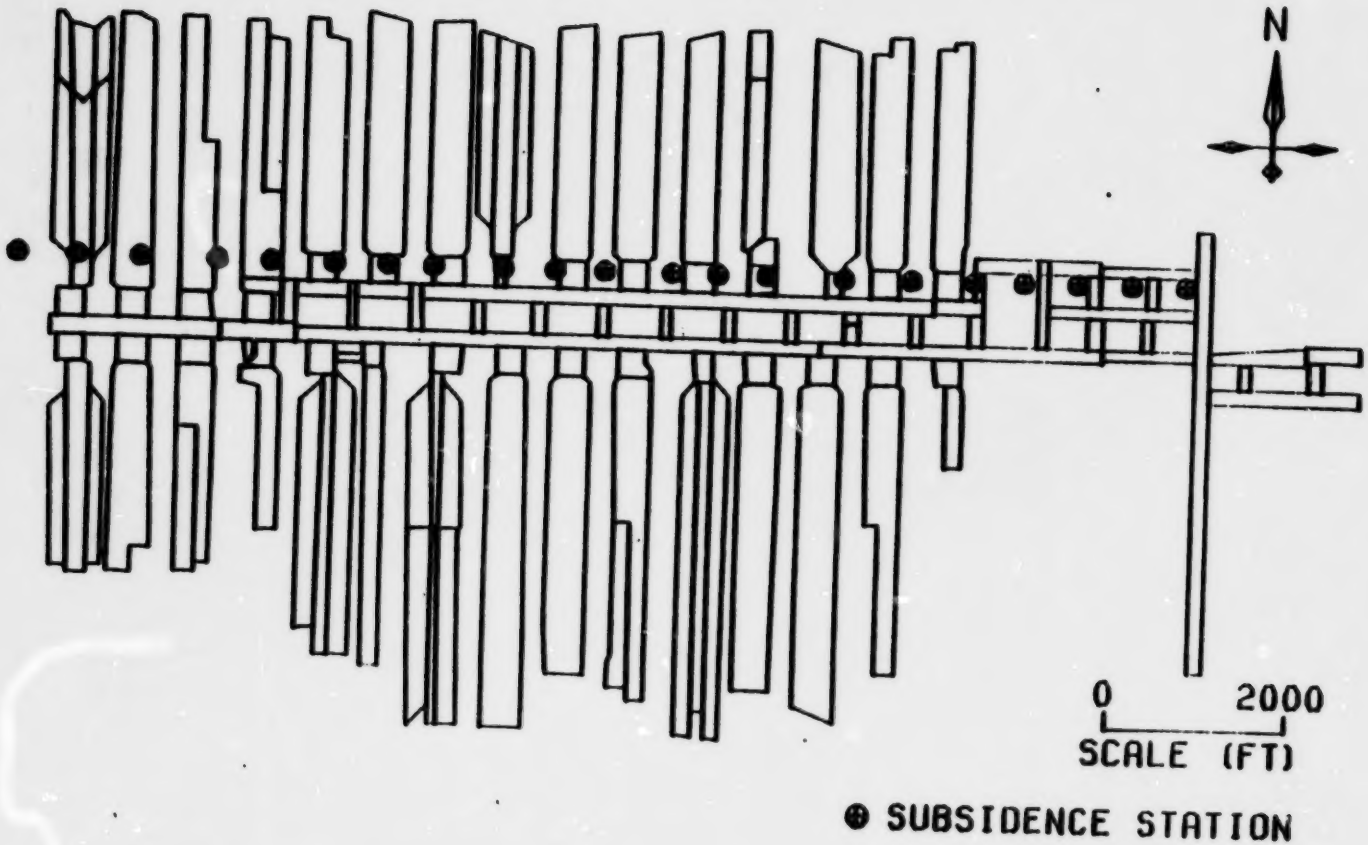


FIG. 32. CCP BLOCK-II WORKINGS AND SUBSIDENCE SURVEY STATION LOCATION

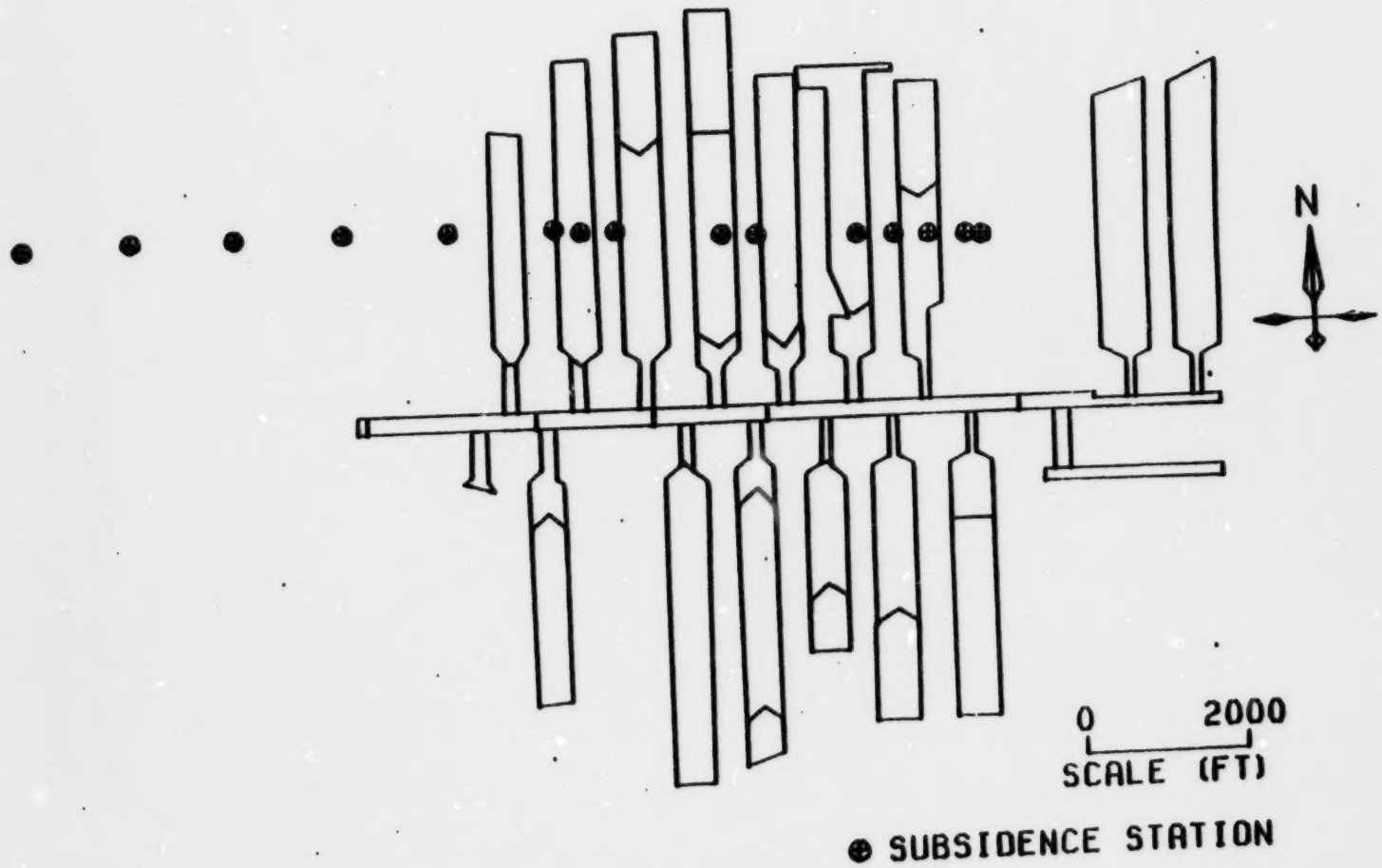
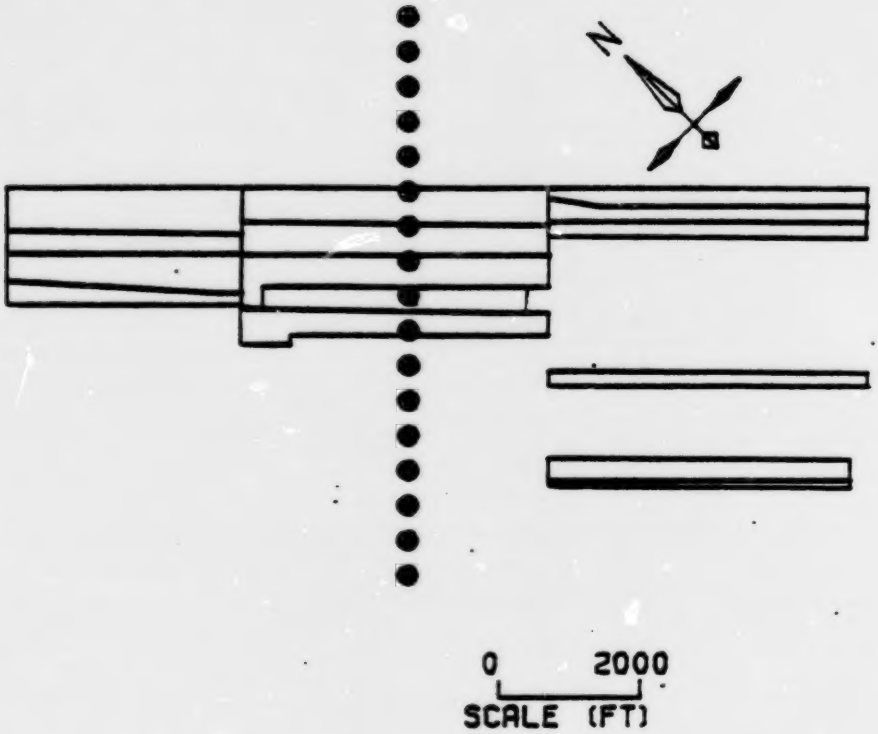


FIG. 33. CORY WORKINGS AND SUBSIDENCE SURVEY STATION LOCATION



⊕ SUBSIDENCE STATION

FIG.34. LANIGAN WORKINGS AND SUBSIDENCE SURVEY STATION LOCATION

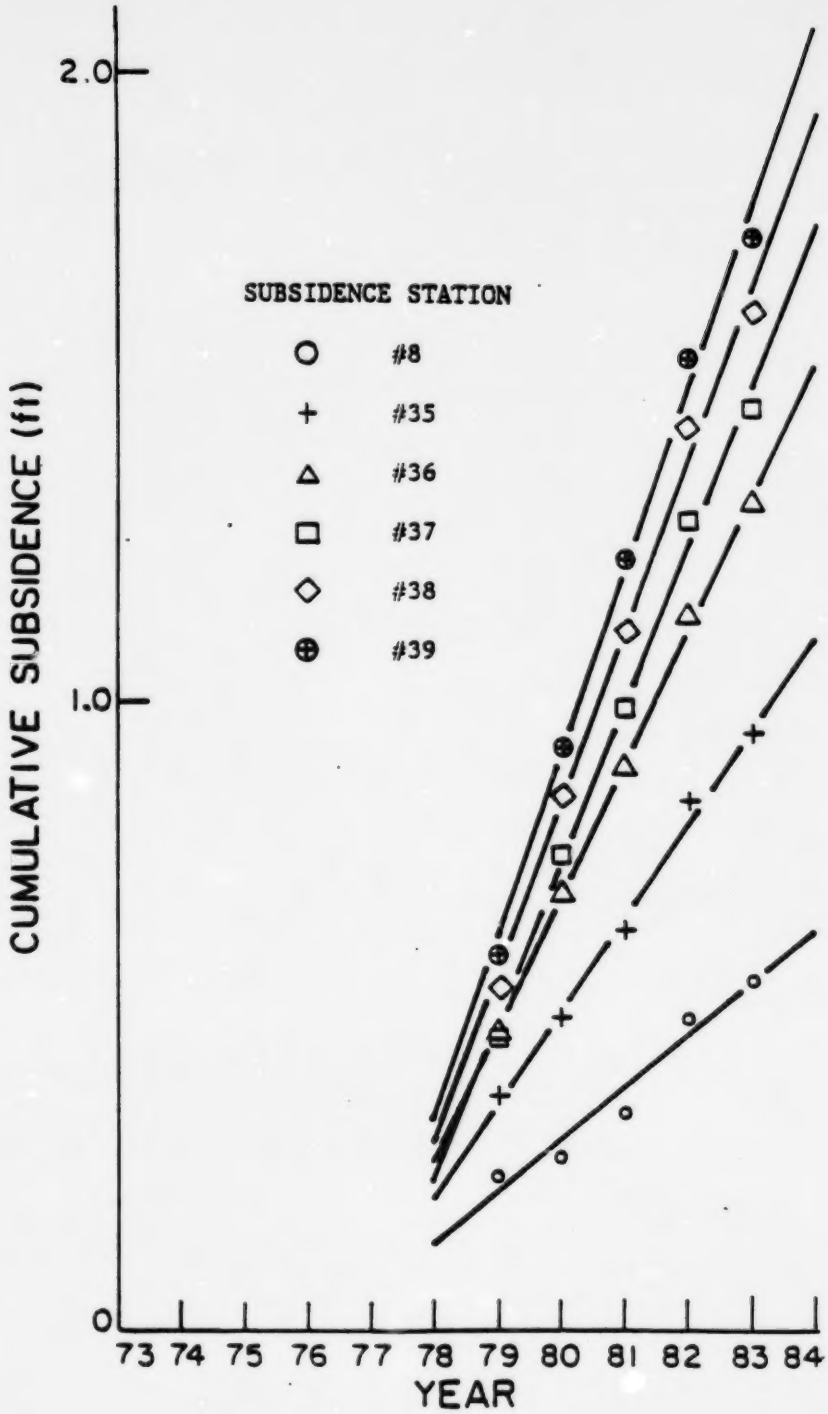


Fig. 35. Subsidence VS. Time for stations lying outside the influence of active mining of CCP Block II workings between 1979 and 1983.

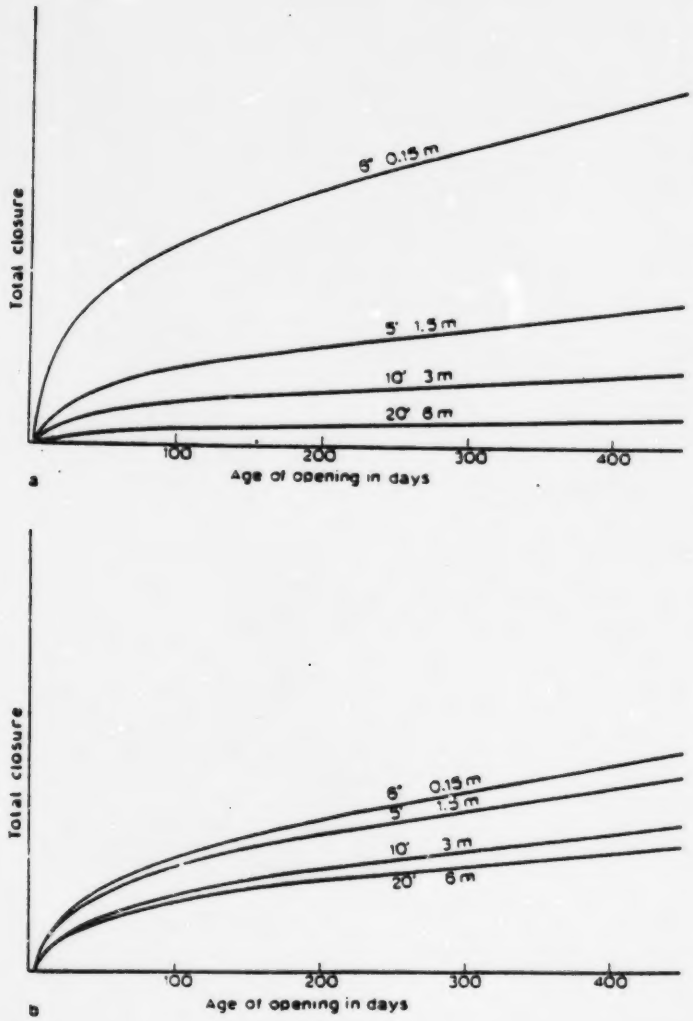


Fig. 36. Closure of the first test room at IMC, measured between points at the indicated distances from the opening.
a. horizontal convergences; b. vertical convergences
(From Baar (4))

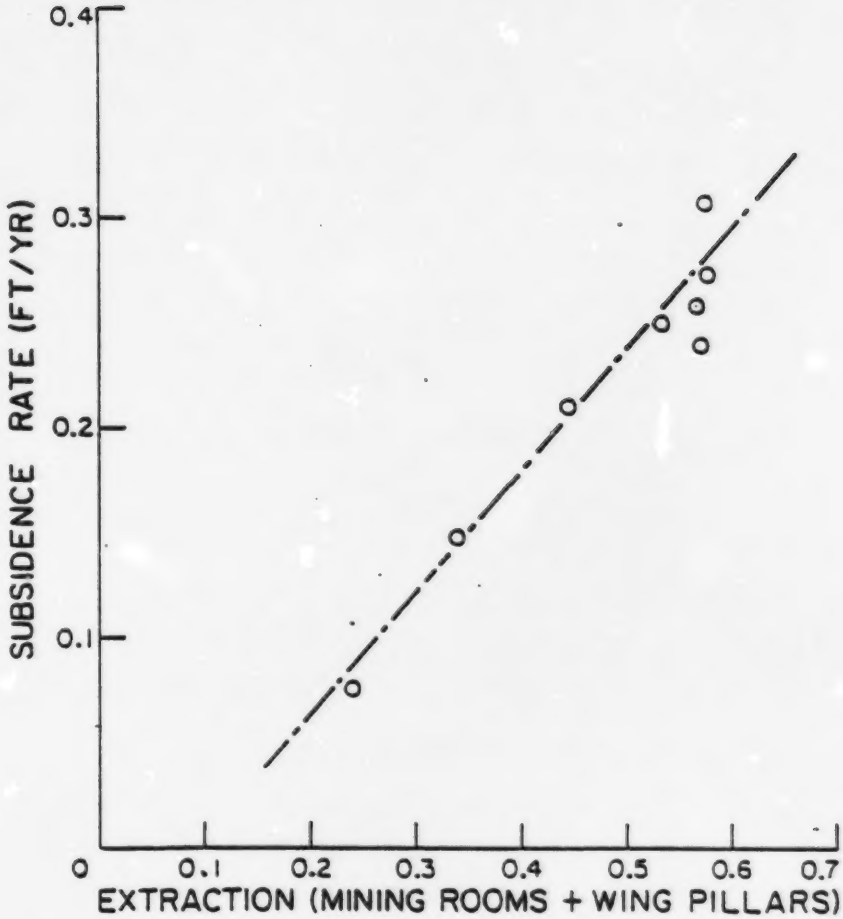


Fig. 37. Comparison of subsidence rate and extraction for selected stations above CCP Block II workings.

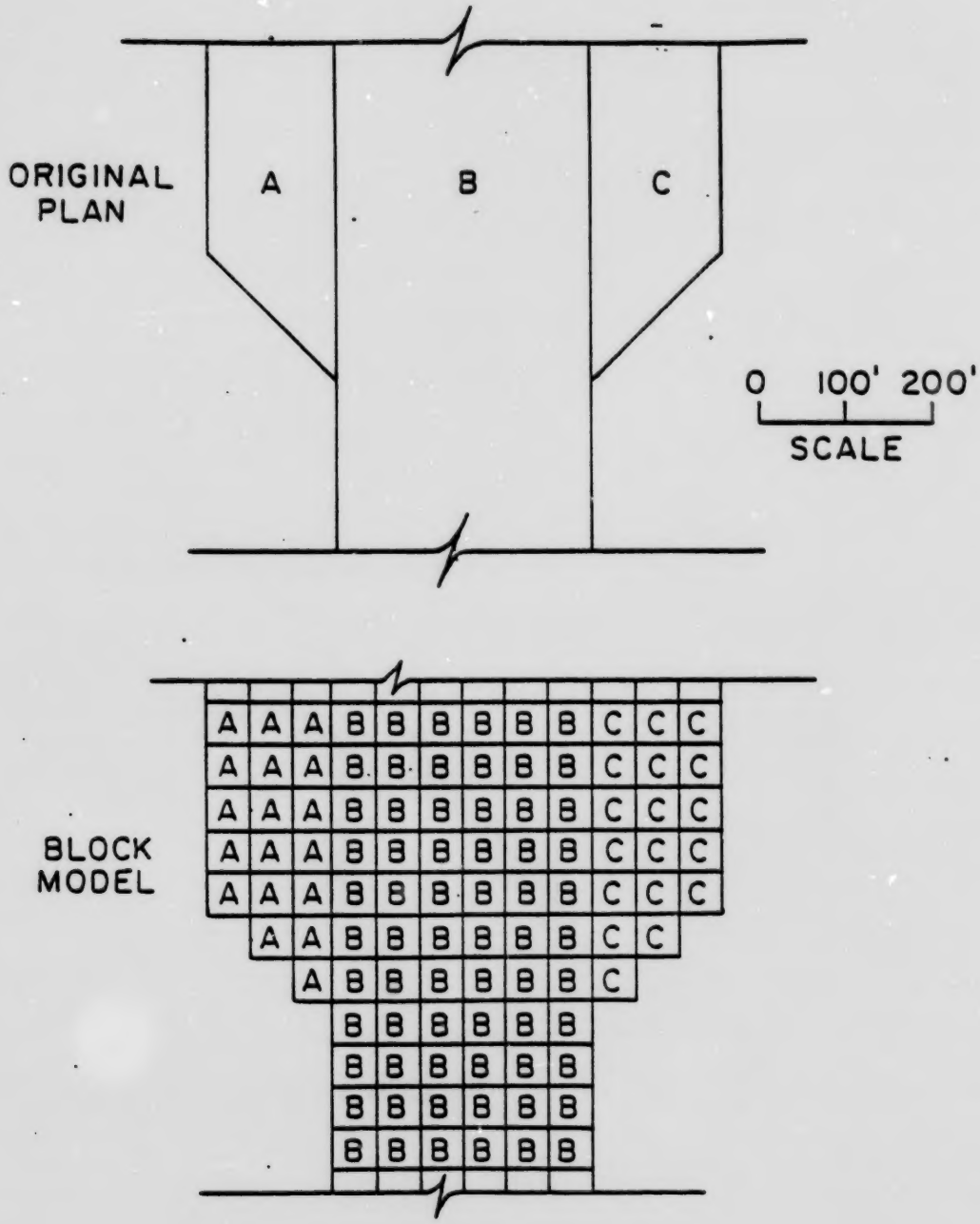


Fig. 38. Representation of mine plan by an area block model. (50'×50' blocks)

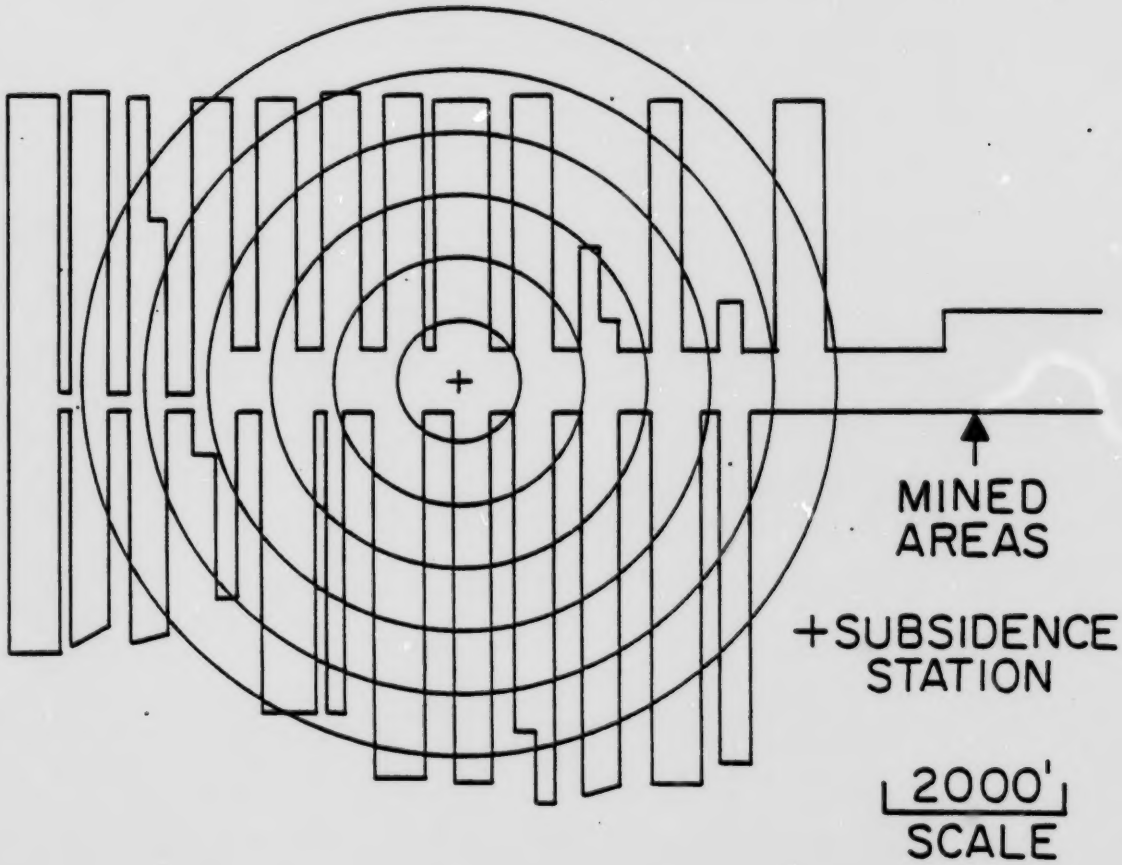


Fig. 39. Area of influence surrounding a subsidence station.

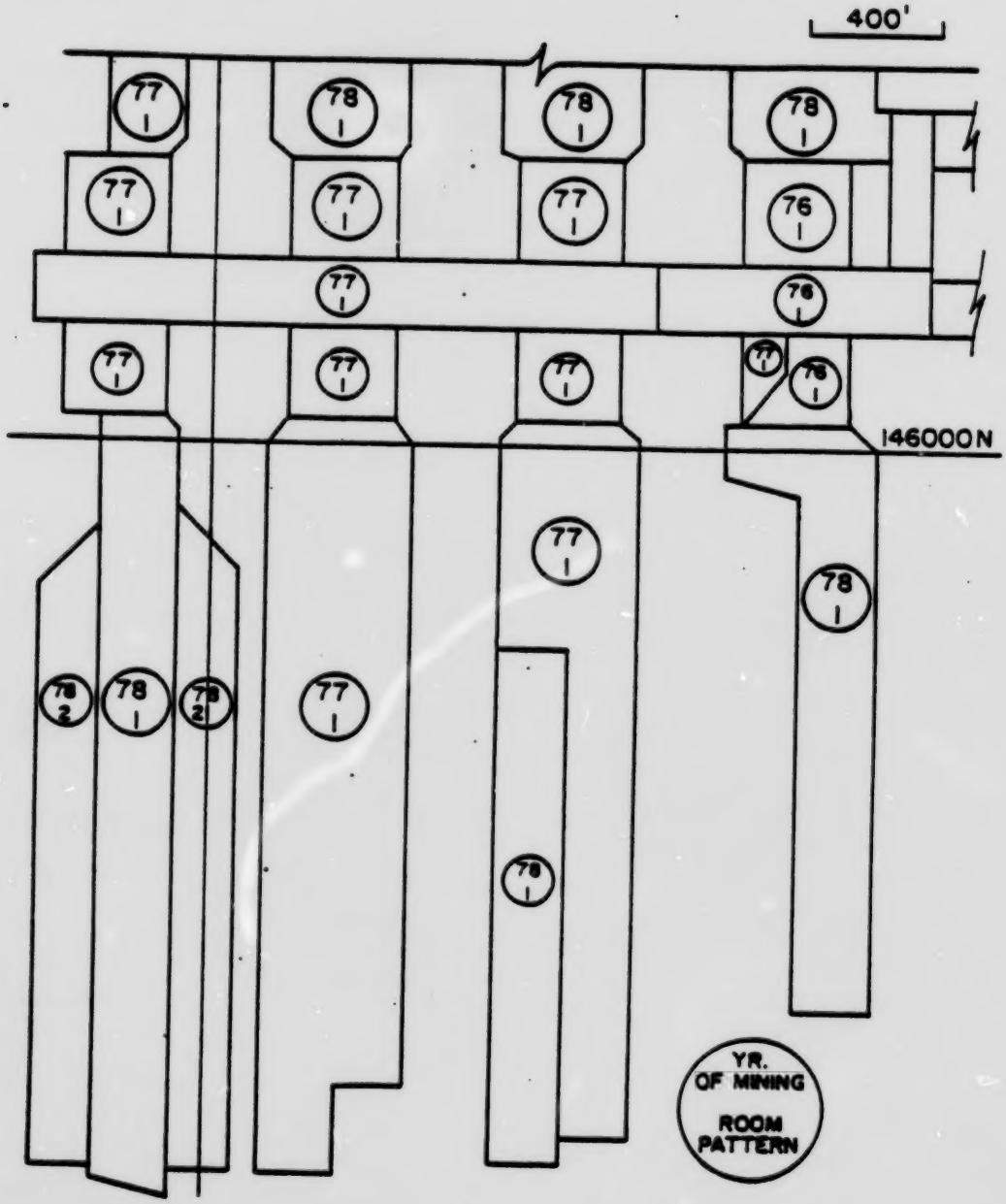


Fig. 40. Simplified plan of workings with year and room pattern indicated.

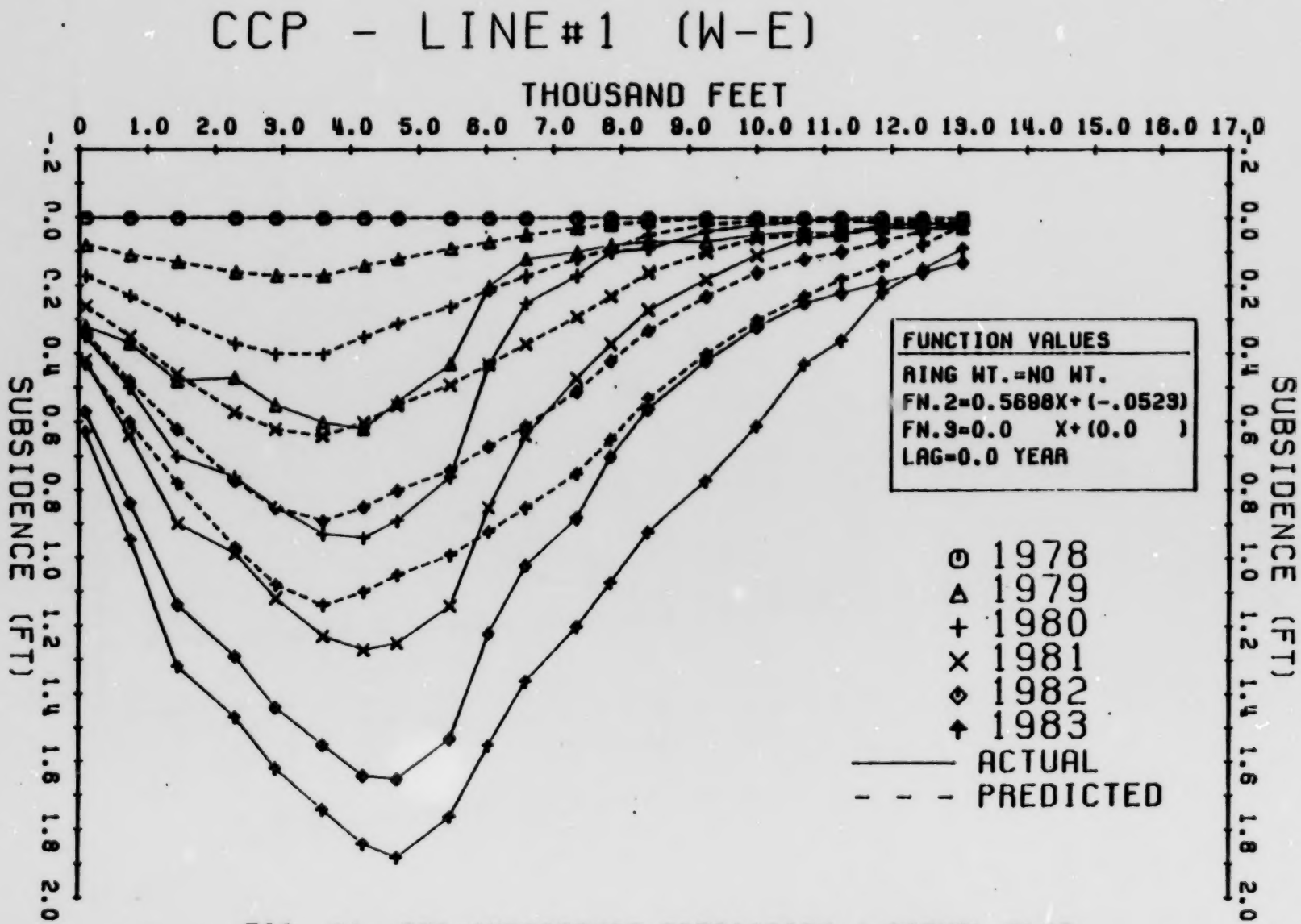


FIG. 41. CCP SUBSIDENCE PREDICTION / ACTUAL PLOT

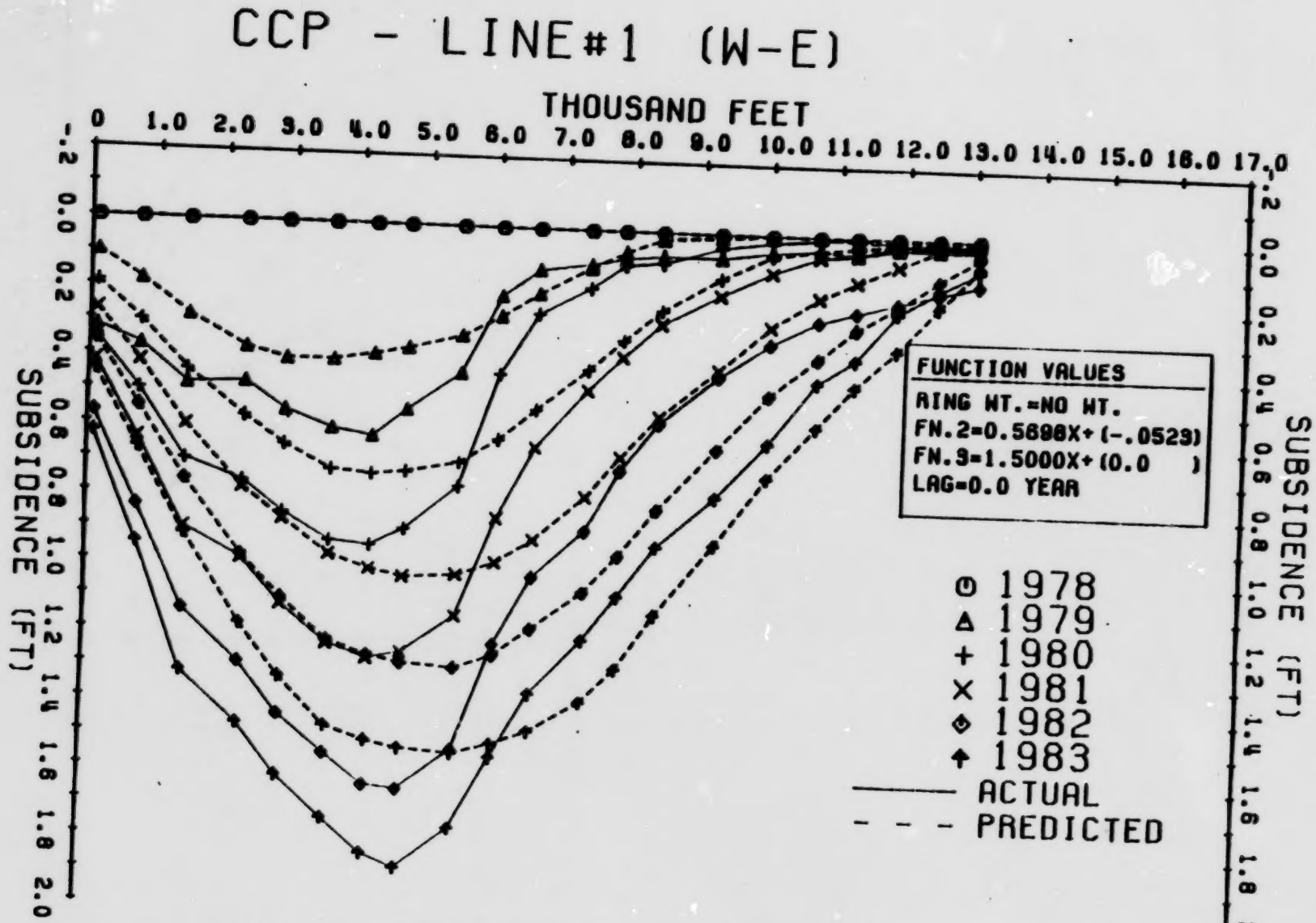


FIG. 42. CCP SUBSIDENCE PREDICTION / ACTUAL PLOT

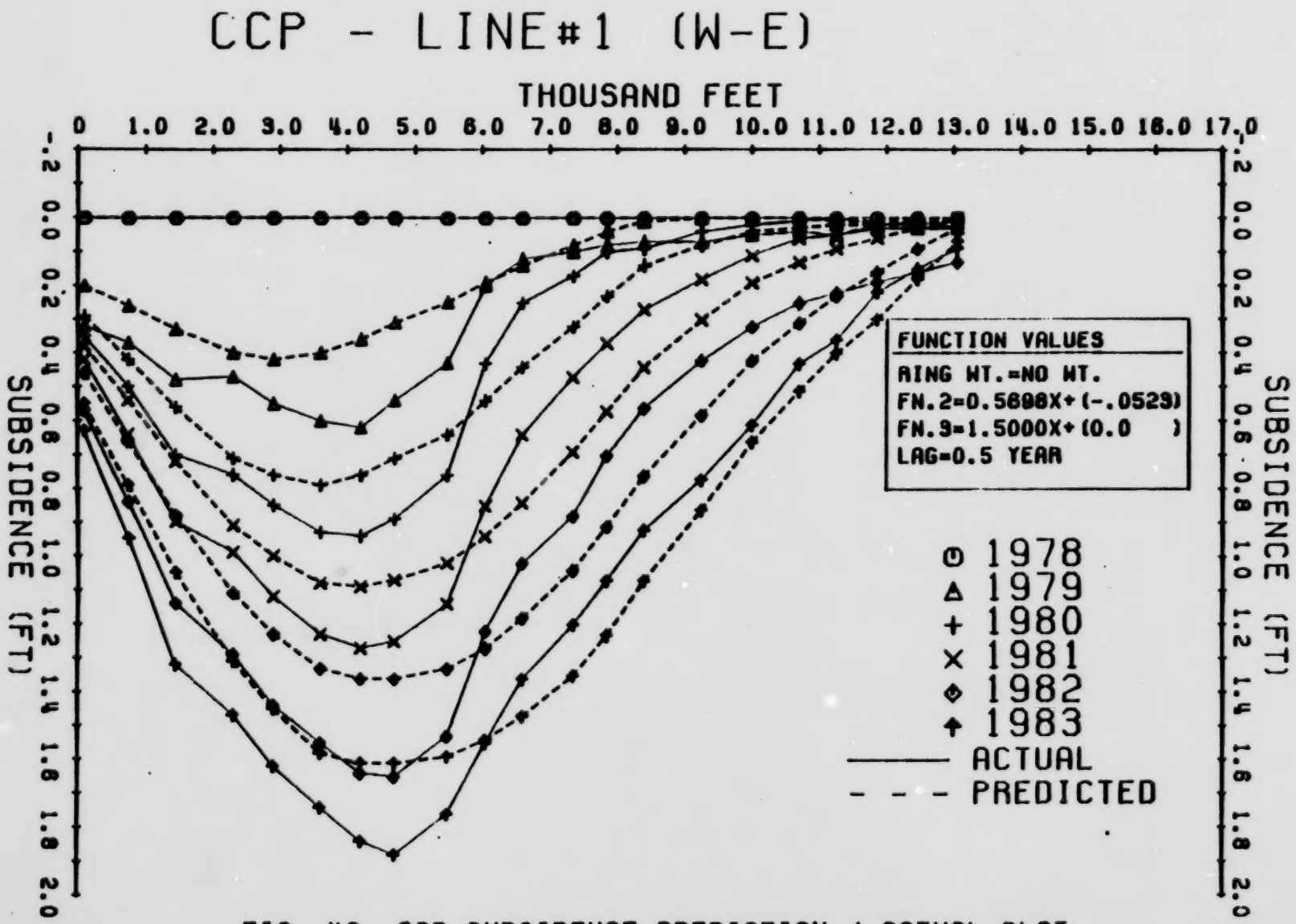


FIG. 43. CCP SUBSIDENCE PREDICTION / ACTUAL PLOT

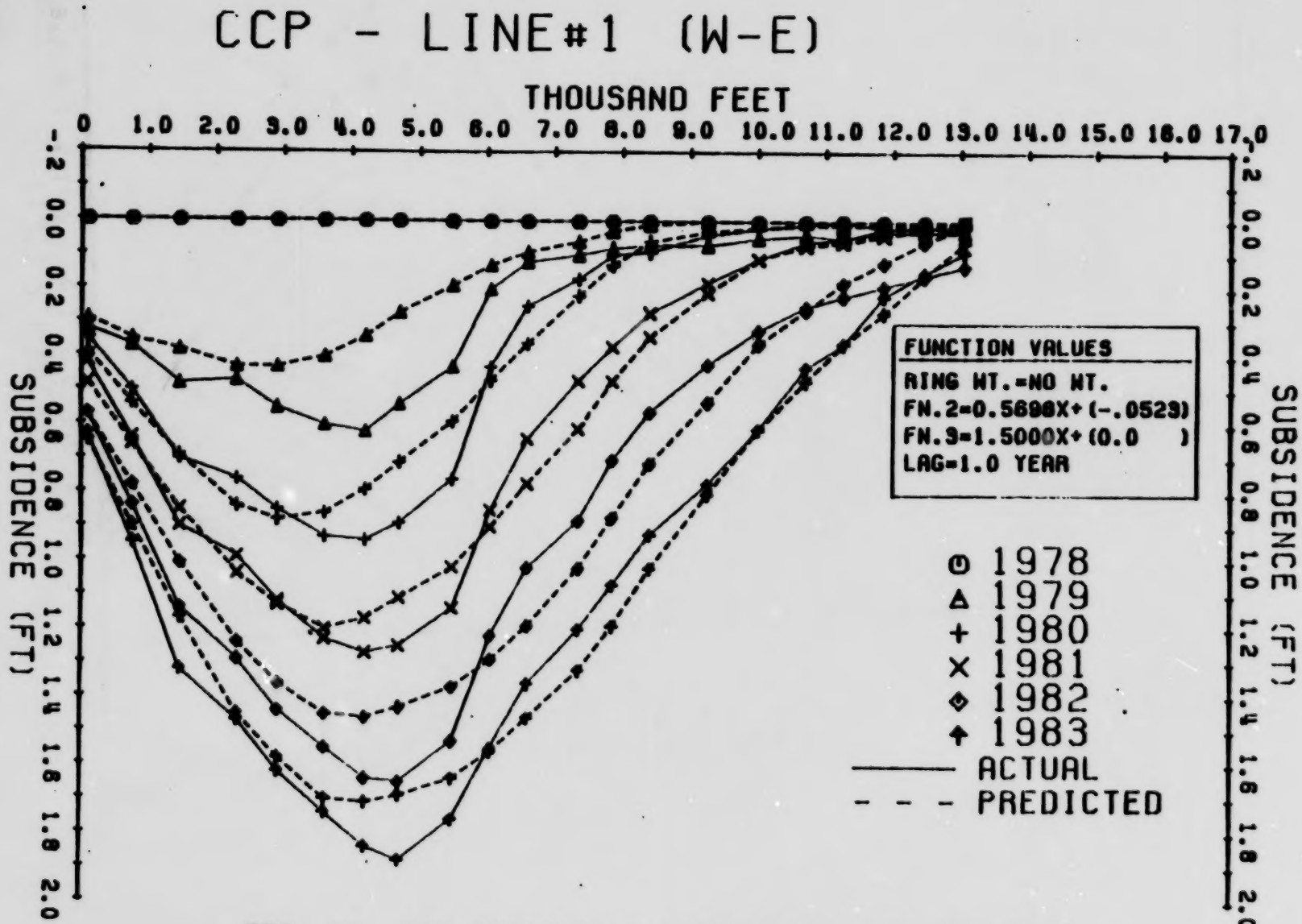


FIG. 44. CCP SUBSIDENCE PREDICTION / ACTUAL PLOT

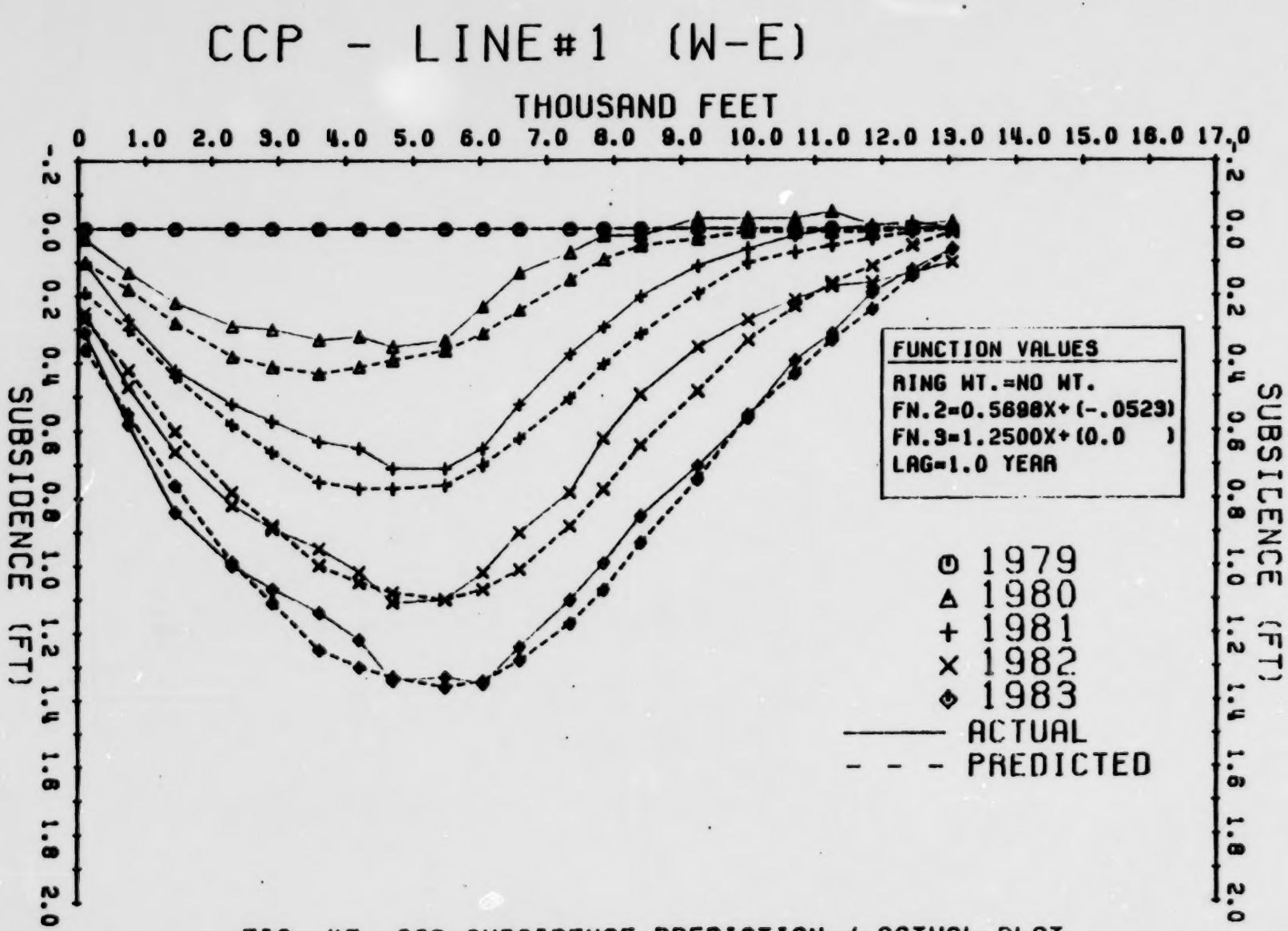


FIG. 45. CCP SUBSIDENCE PREDICTION / ACTUAL PLOT

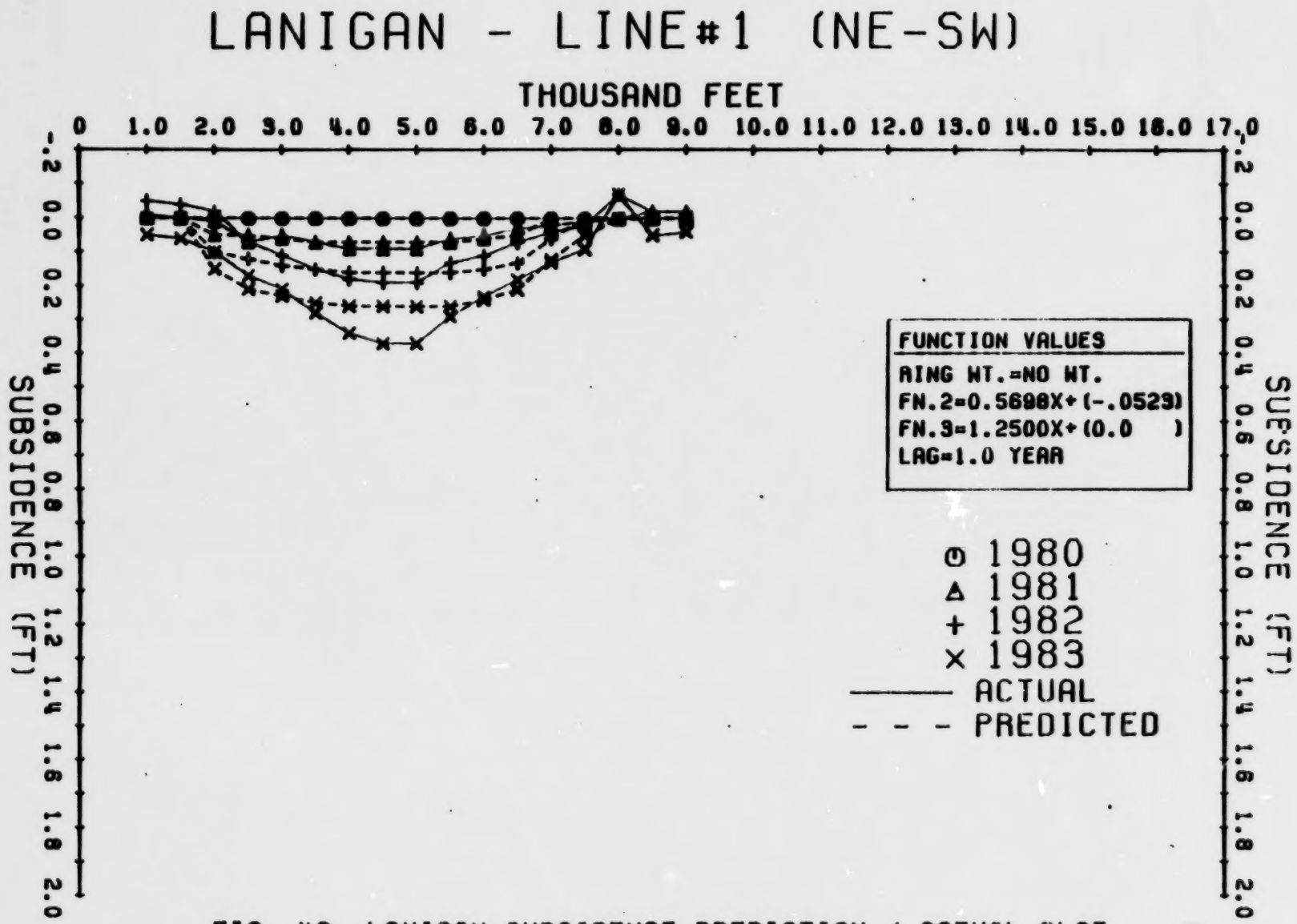


FIG. 46. LANIGAN SUBSIDENCE PREDICTION / ACTUAL PLOT

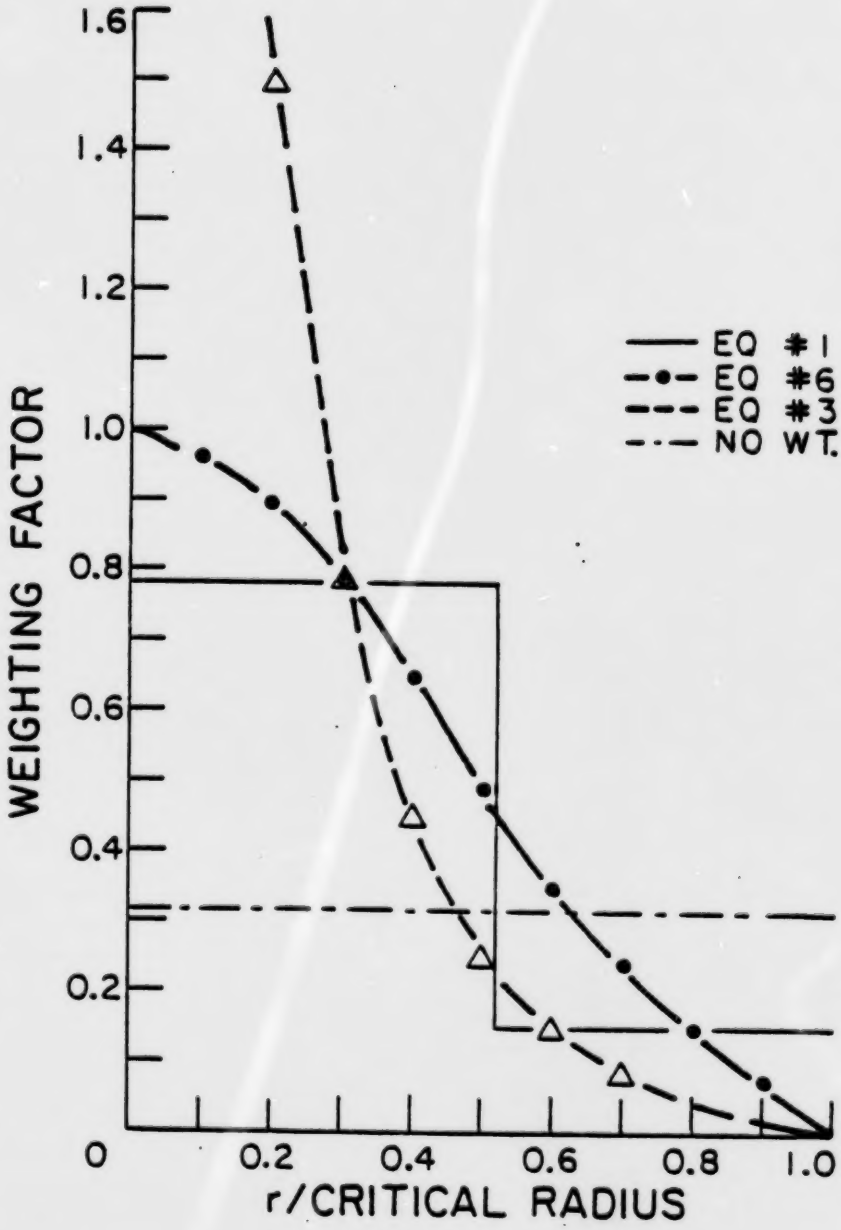


Fig. 47. Plotted influence functions (From Table 5)

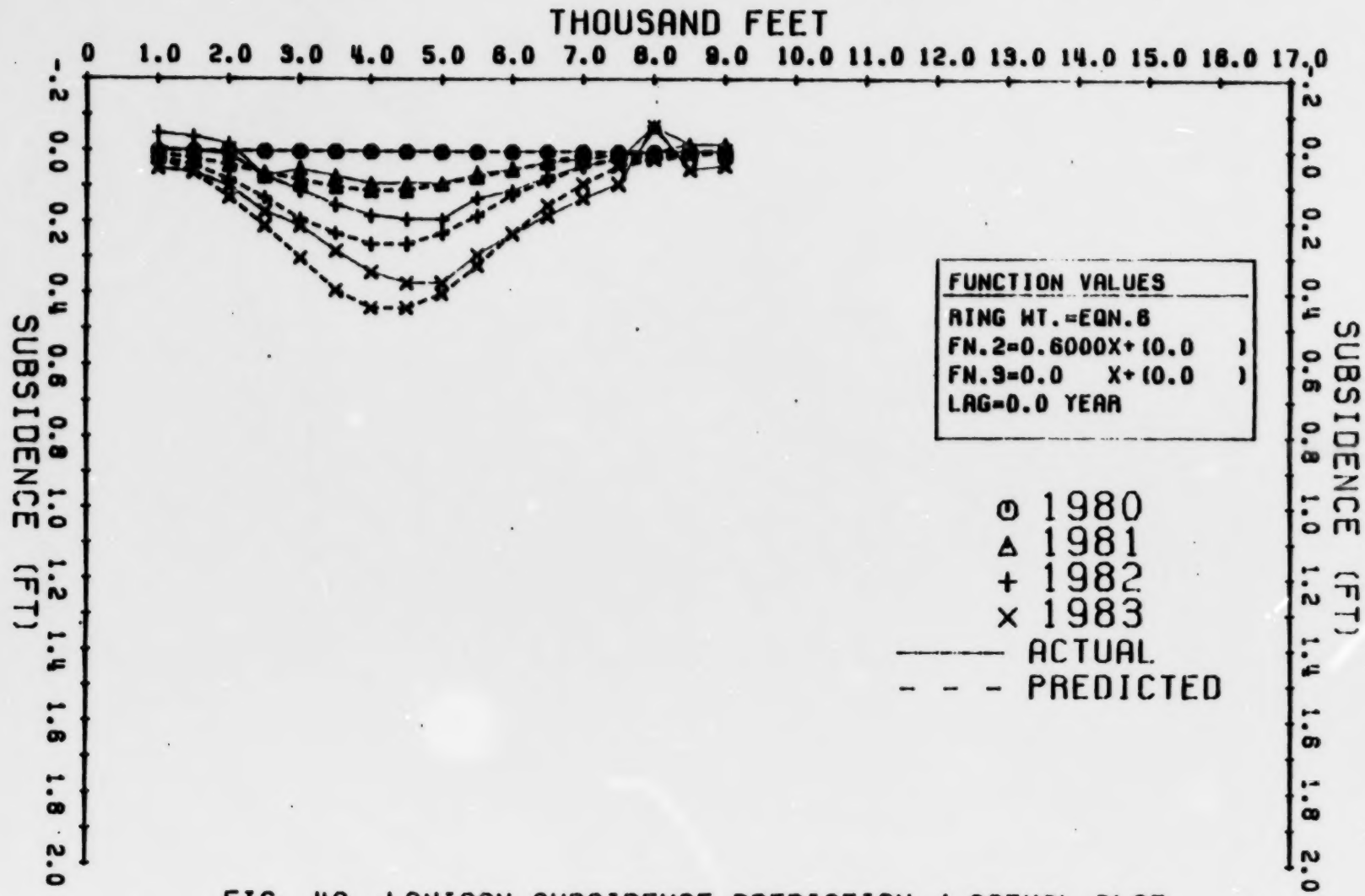


FIG. 48. LANIGAN SUBSIDENCE PREDICTION / ACTUAL PLOT

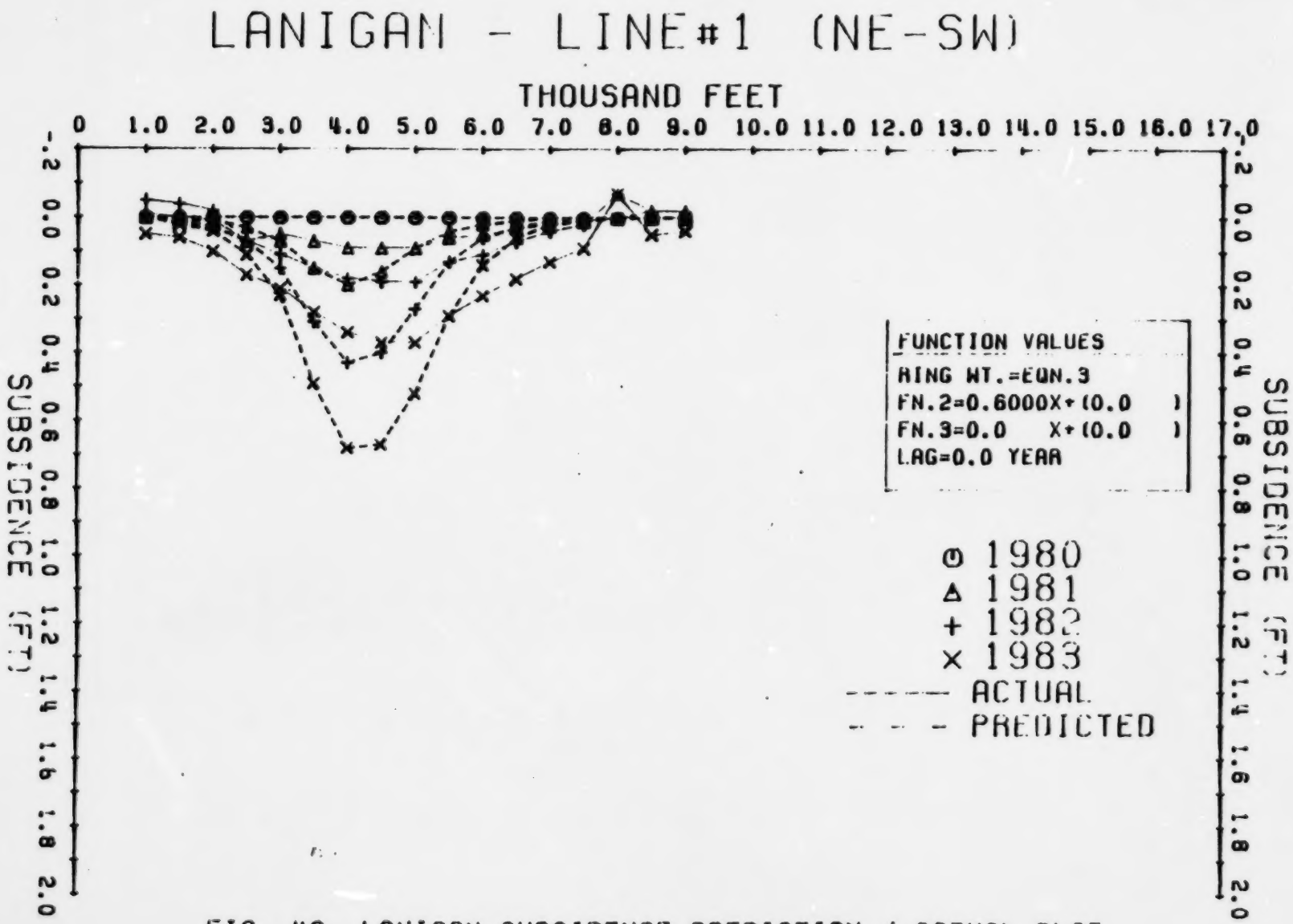


FIG. 49. LANIGAN SUBSIDENCE PREDICTION / ACTUAL PLOT

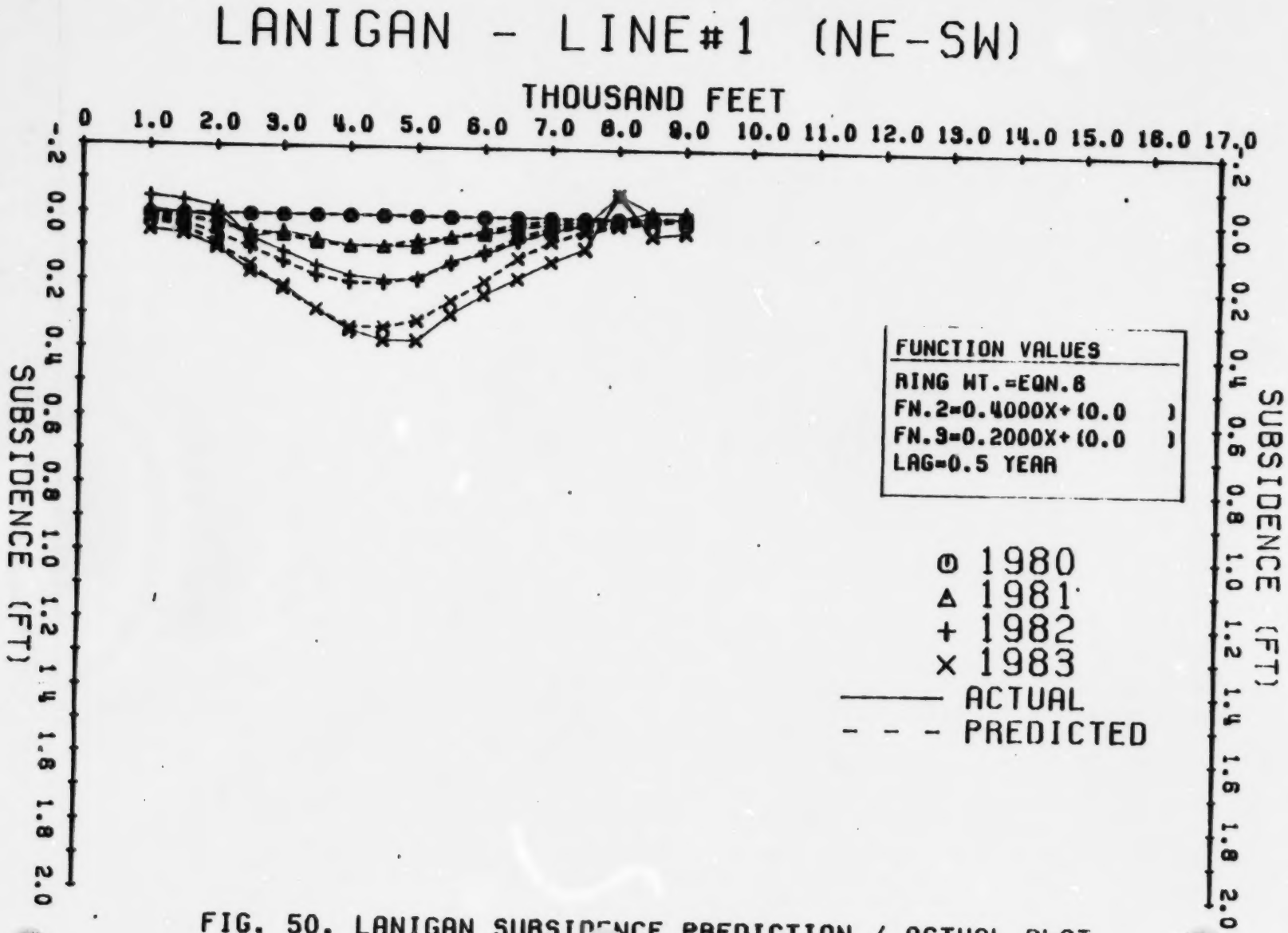


FIG. 50. LANIGAN SUBSIDENCE PREDICTION / ACTUAL PLOT

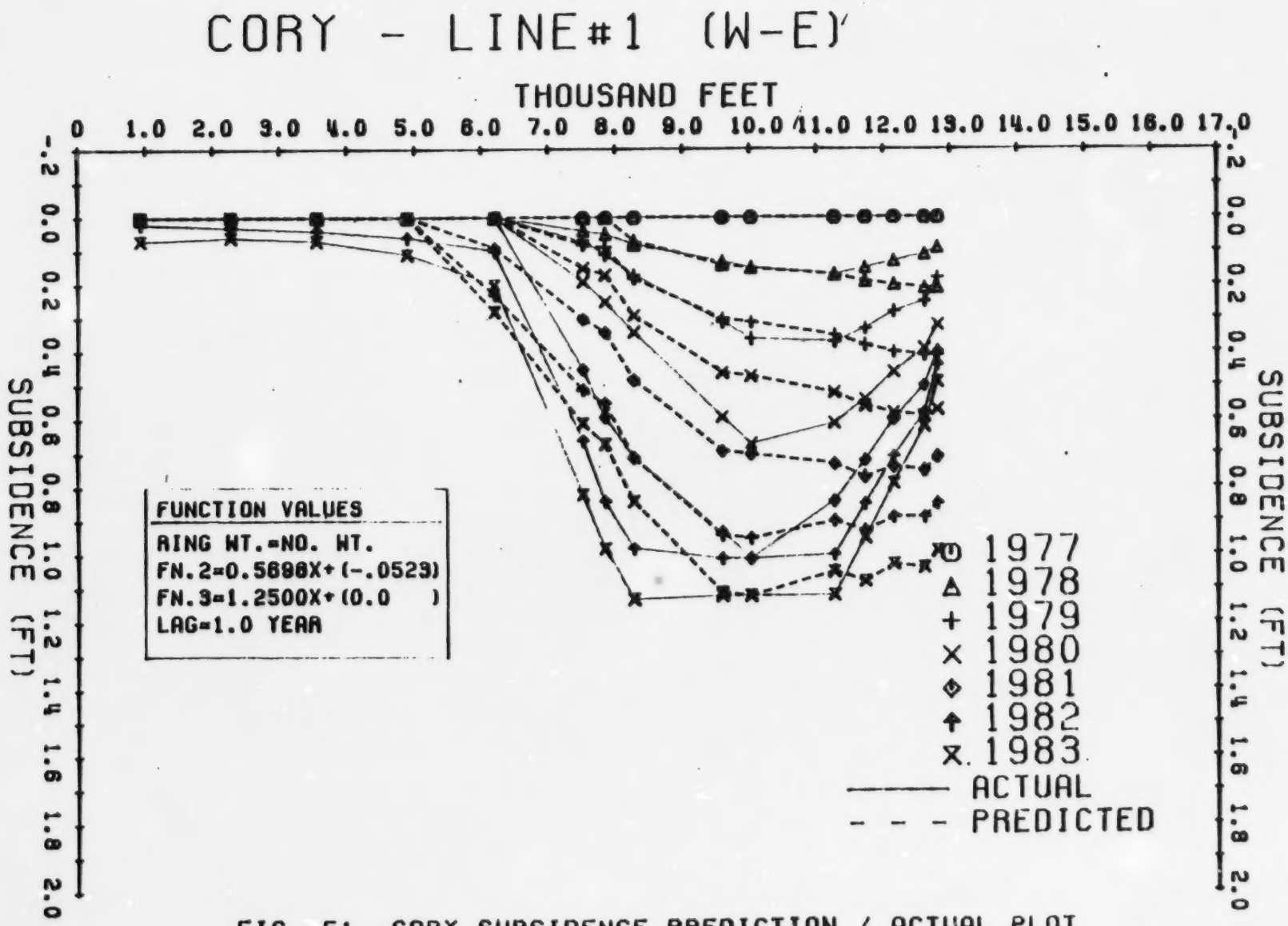


FIG. 51. CORY SUBSIDENCE PREDICTION / ACTUAL PLOT

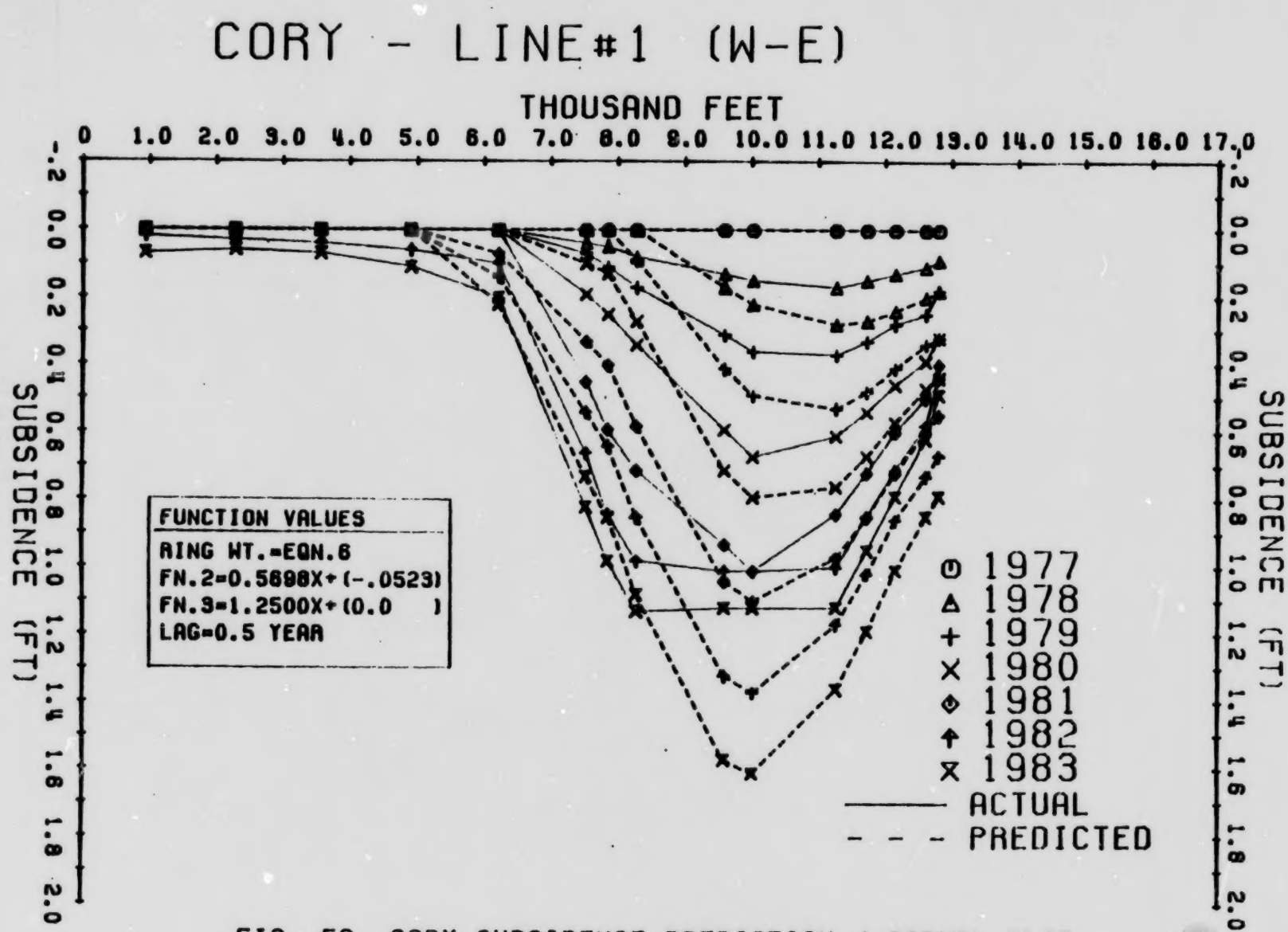


FIG. 52. CORY SUBSIDENCE PREDICTION / ACTUAL PLOT

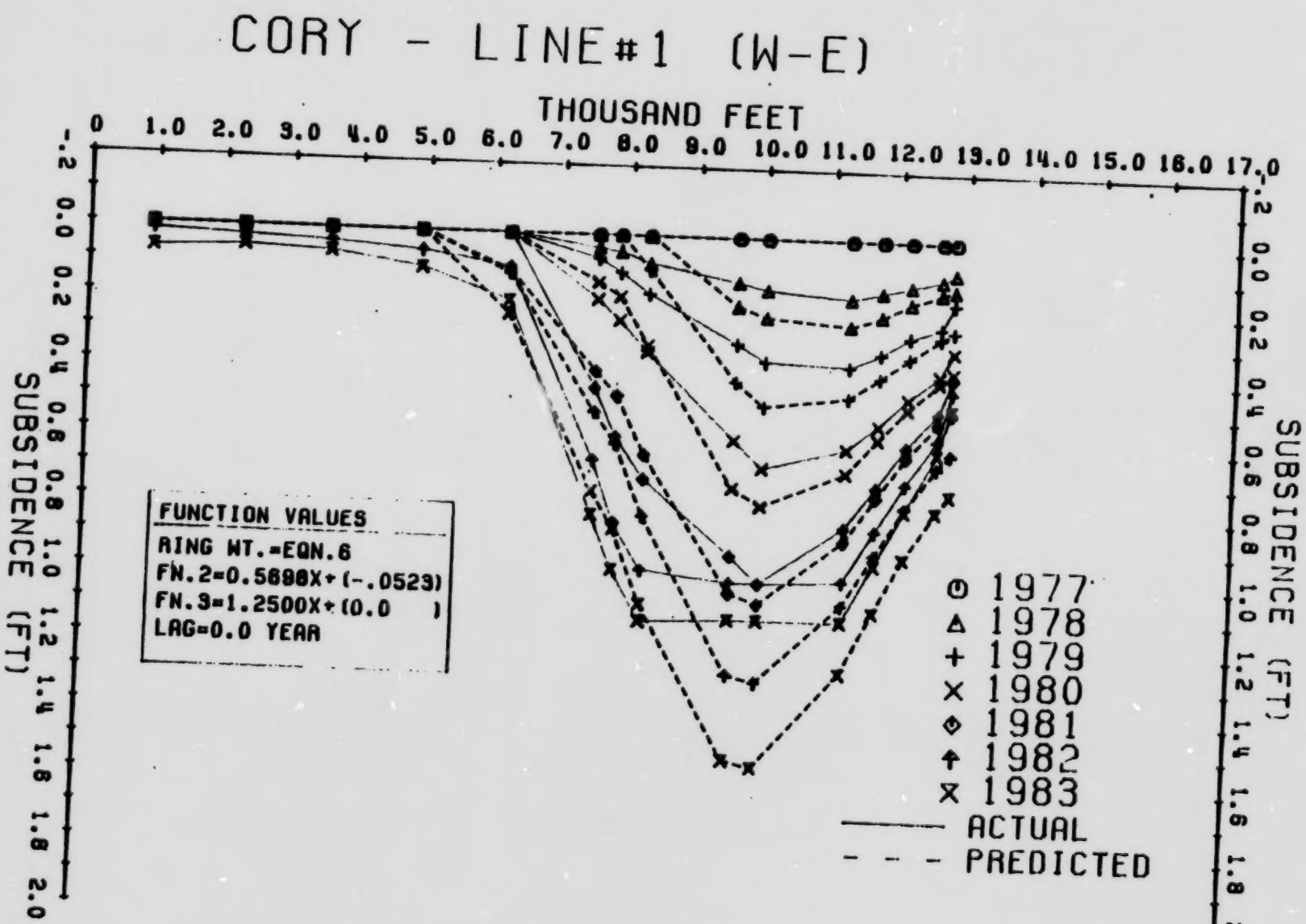


FIG. 53. CORY SUBSIDENCE PREDICTION / ACTUAL PLOT

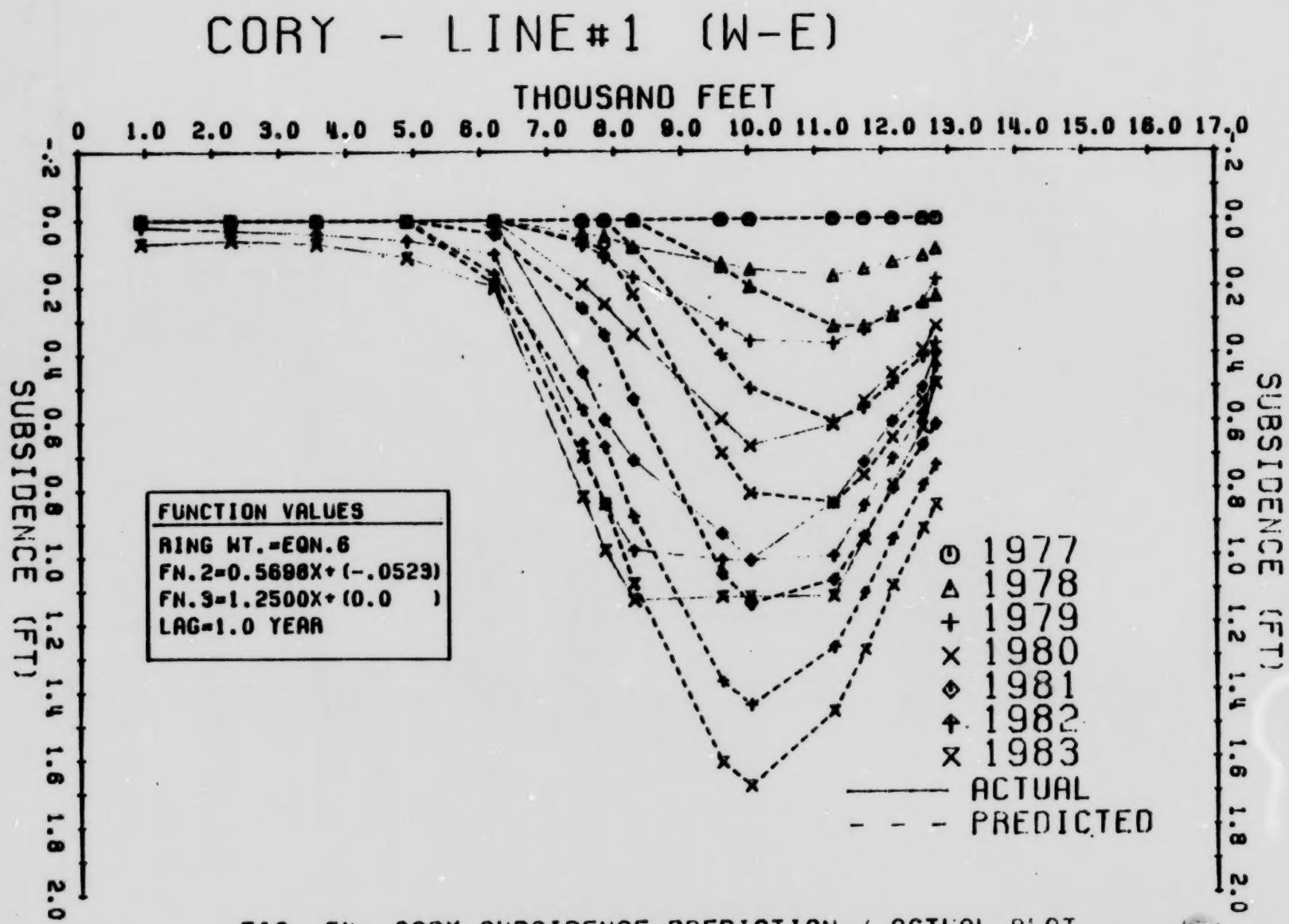


FIG. 54. CORY SUBSIDENCE PREDICTION / ACTUAL PLOT

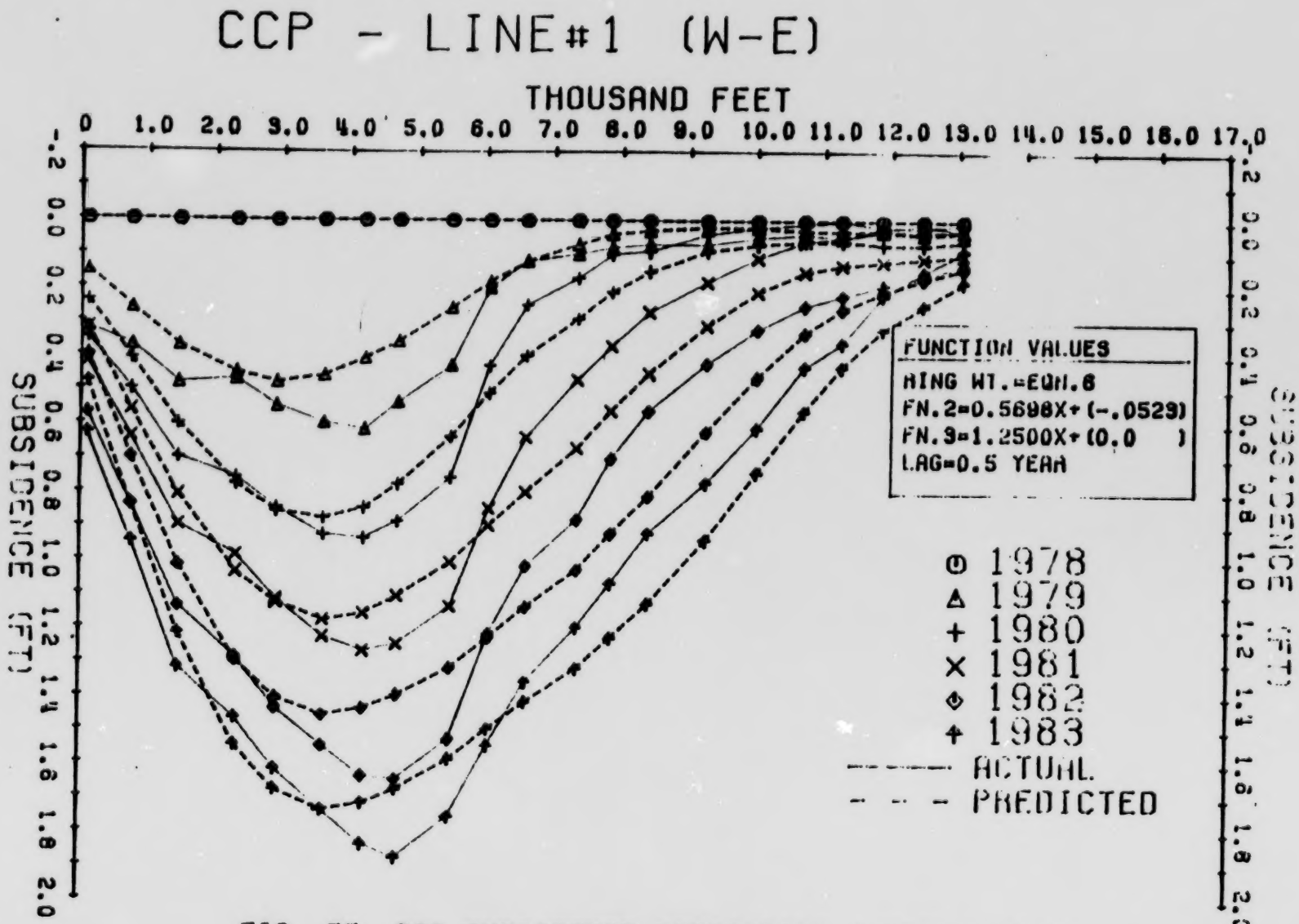


FIG. 55. CCP SUBSIDENCE PREDICTION / ACTUAL PLOT

TABLE 1.

SUBSIDENCE FACTORS

(from Braeuner (11))

Coalfield and method of packing	a
British coalfields:	
Solid stowing	0.45
Caving or strip-packing	.90
Ruhr coalfield, Germany:	
Pneumnatic stowing	.45
Other solid stowing	.50
Caving	.90
North and Pas de Calais coalfield, France:	
Hydraulic stowing	0.25-0.35
Pneumatic stowing	.45- .55
Caving	.85- .90
Upper Silesia:	
Hydraulic stowing	.12
Caving	.70
U.S.S.R.[1]:	
Donets and Kizelov districts	.60
Kuznetsk and Karaganda districts	.70
Chelybinsk district	.90

[1]Mining without stowing, which implies caving.

Observations in American coalfields indicate rather small subsidence factors for caving. From measurements in Pennsylvania, values between 0.50 and 0.60 can be derived.

TABLE 2

COALFIELD LIMIT ANGLES

(from Braeuner (11))

Limit Angle (degrees)	Field
45-55	Limburg, Netherlands
45-60	Ruhr, Germany
55	N. France
60	Soviet Union
55-65	Great Britain
70	Pennsylvania, USA

TABLE 3

PROFILE FUNCTIONS

(from Braeuner (11))

Profile Function		Area of Application
1)Hyperbolic	$s(x)=0.5\times S\times(1-\tanh(bx/R)$	U.K.
2)Error	$s(x)=0.5\times S\times(1-(2/\sqrt{\pi})\int_0^{\pi x/b} e^{-u^2}du)$	Upper Silesia Poland
3)Exponential	$s(x)=S\times\exp(-(1/2)[(x+b)/R]^2$	Hungary
4)Donets Coal Field	$s(x)=0.5\times S(1-(x/R)-(1/\pi)\sin\pi(x/R))$	USSR
5)Trigonometric	$s(x)= S\times\sin^2[(\pi/4)(x/R-1)]$	

TABLE 4

PROFILE FUNCTIONS OVER SUBCRITICAL AREAS

(from Braeuner (11)

1)DONETS COALFIELD

a)for $N>1/3$

$$s(x)=S[N^2(1-X_e+(\sin 2\pi X_e)/2\pi)+(1-N^2)/4\times(1+\cos X_e\pi)]$$

b)for $N<1/3$

$$s(x)=S[(1+\cos X_e\pi)^2]$$

where:

$S = Smax[(N1\times N2)^2]$
$X_e = x/L$
$N1 = W1/(2B)$
$N2 = W2/(2B)$
W1 = subcritical width #1
W2 = subcritical width #2
N is in the direction parallel to the profile
If $W>2\times B$ then set $N=1$

2)SUPERPOSITION OF HYPERBOLIC FUNCTIONS

$$s(x)=0.5(S)[\tanh[2(x+W)/R]-\tanh(2x/R)]$$

where:

x = 0,is over the right ribside
W = subcritical width
$S = Smax\times\tanh(W/R)$
= maximum subsidence occurs at $x=-0.5W/R$

TABLE 5

INFLUENCE FUNCTIONS

(from Braeuner (11))

1) $p(r)=[(S)(R^3\tan^3\beta))]/\pi[[\sin\beta\cos\beta+(\pi/2)-\beta)][r(r^2+R^2\tan^2\beta)]^2]$,

where: β=Break angle

2) $p(r)=3(S)[1-(r/R)^2]^2/\pi R^2$

3) $p(r)=2(S)\exp[-4(r/R)^2]/(\pi^{1.5}Rr)$

4) $p(r)=0.216(S)\exp[-4(r/R)^6]/Rr$

5) $p(r)=(S)/\exp[-(r/R)^2]/R^2$

6) $p(r)=4.6(S)/\exp[-4.6(r/R)^2]/\pi R^2$

7) $p(r)=2(S)\exp(-2(r/R)^2]/R^2$

8) $p(r)=7(S)/\exp(-6.65\times r/R)/R^2$

TABLE #6

ANGLE OF DRAW OBSERVED FROM SUBSIDENCE DATA

MINE	OBSERVED ANGLE OF DRAW *
ALLAN	Questionable survey data
CCP	46°-54°
CORY	48°-55°
LANIGAN	54°-68°
ROCANVILLE	≈68°

* angles measured from the horizontal

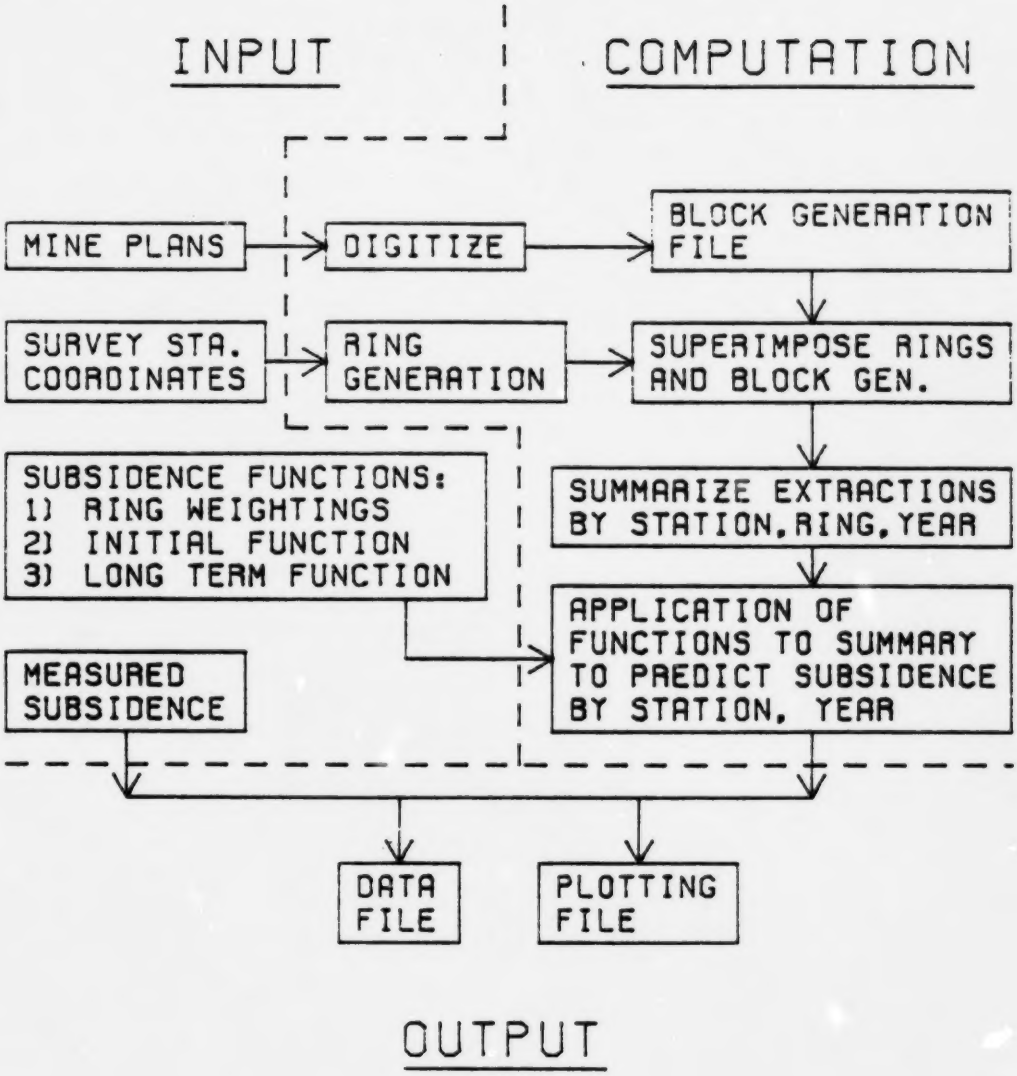


TABLE #7
FLOW CHART OF SUBSIDENCE PREDICTION COMPUTER PROGRAM

APPENDIX 1

TYPICAL SECTION THROUGH WORKINGS, ILLUSTRATING STANDARD SYMBOLS FOR SUBSIDENCE, SLOPE, STRAIN, AND DISPLACEMENT.
(From N.C.B. handbook (52))

Section 1: Symbols, Terms and Definitions

Many of the terms in the following list are used only in research and not in everyday subsidence engineering work, but they are included because they are all closely associated with factors in everyday use.

Some strata control terms are also included to cover the liaison between the two disciplines.

Figs. 1 and 2 illustrate some of the symbols which are listed.

Other symbols, i.e. those used in the interpretation of mine plans and geological plans, will be found in the Board's Code of Surveying Practice.

Symbols	Terms	Definitions
	Abutment	The region adjoining an excavation which is normally subjected to increased stress as a consequence of the redistribution of load. This may be at the front of, or at the sides or rear of, a working face.
	Abutment pressure	The load per unit area on an abutment.
β	Angle of break	The angle of inclination of the line connecting the edge of the workings and the point of maximum lengthening.
α	Angle of dip	The apparent dip of the seam.
γ	Angle of draw (limit angle)	The angle of inclination from the horizontal of the line connecting the edge of the workings and the edge of the subsidence area.
ζ	Angle of draw (limit angle)	The angle of inclination from the vertical of the line connecting the edge of the workings and the edge of the subsidence area.
	Arch core	The relaxed strata within a pressure arch.
	Block	An area of coal worked between predetermined boundaries.
	Convergence	Movement of roof and floor towards each other after removal of mineral. The rate of convergence is measured as either: (1) the convergence for a given advance of the face; or (2) the convergence in a given time.

Symbols	Terms	Definitions
	Convergence (initial)	The amount of convergence of the roof and floor in the face directly after working and prior to stowing.
	Cover (see also *depth*)	The strata lying between the surface and the seam.
	Cover load	The dead load of the cover.
	Critical Area	That area the working of which causes the complete subsidence of one point on the surface.
	Sub-critical area:	An area of working smaller than the critical area. (When a sub-critical area is being worked, the point on the surface under examination does not undergo complete subsidence.)
	Super-critical area:	An area of working greater than a critical area. (An area on the surface undergoes complete subsidence.)
R	Critical area (radius of)	
ρ	Curvature (radius of)	The radius of curvature of any part of a subsidence profile.
h	Depth	The vertical distance between the floor of the seam being extracted and any arbitrary datum – usually the surface.
v	Displacement	The horizontal displacement of a point caused by any working.
V	Displacement (maximum)	The maximum horizontal movement of a point caused by working a sub-critical area.
V_{max}	Displacement (maximum possible)	The maximum horizontal movement of a point caused by working a critical area.
$v_x v_y$	Displacement in the direction of co-ordinate axes.	The partial displacement in the direction of the axes of a co-ordinate system, on which the calculations are based.

Symbols	Terms	Definitions
v_t	Displacement (spatial)	The rectilinear connection of the initial and final positions of a point moving in space. (It is determined by direction, inclination and distance.)
d	Distance	Any horizontal distance.
	Face	1. Strictly, any surface exposed by excavation for development or for the getting of mineral; 2. More generally, the supported area in the vicinity of the place at which mineral is worked.
	Face length	The distance between rib-sides or goaf-sides of a coal face.
	Face width	The distance between the line of coal face and the waste edge.
	Panel	The area of coal extracted by a longwall face.
L	Panel length	The dimension of a panel measured in the direction of the line of face advance.
w	Panel width	The transverse distance across a panel, usually equal to the face length.
	Pillar	A block of coal or mass of ore left unworked.
p	Pillar width	The shortest dimension of a rectangular pillar.
	Pressure arch	A zone of increased stress surrounding the relaxed strata which is created when any excavation is made, resulting in the redistribution of the normal strata pressures on to the abutments.
	Profile (longitudinal)	A curve depicting subsidence on a section drawn parallel to the direction of advance of an underground excavation.

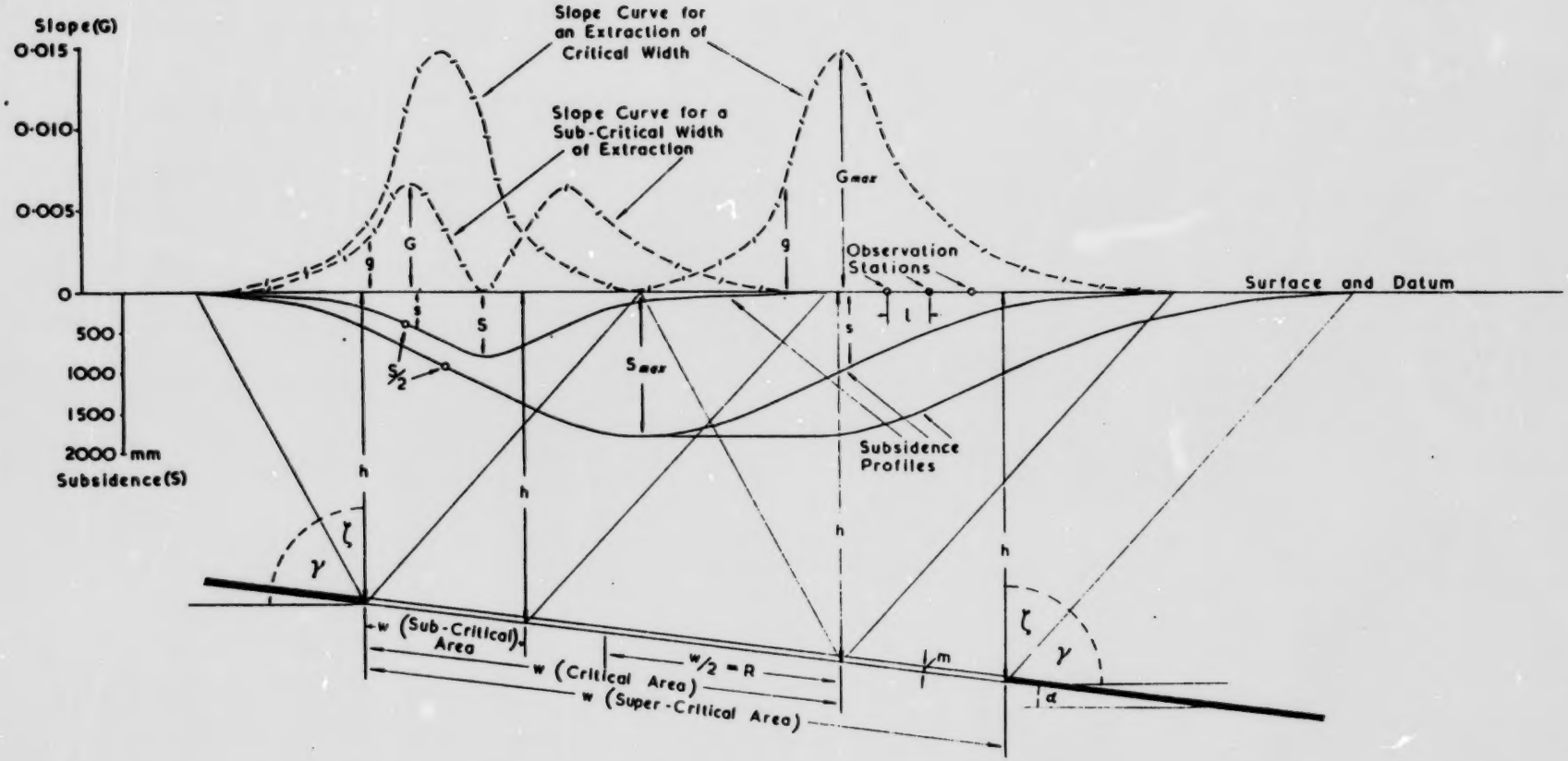


Fig. 1 Typical section through workings, illustrating standard symbols for subsidence and slope.

Symbols	Terms	Definitions
	Profile (transverse)	A curve depicting subsidence on a section drawn at right-angles to the direction of advance of an underground extraction.
g	Slope	The slope of any part of a subsidence trough. NOTE: All forms of slope are calculated as the difference in subsidence between two points close together divided by their distance apart, and expressed in parts per thousand.
G	Slope (maximum)	The maximum slope in a subsidence trough caused by working a sub-critical area.
G_{max}	Slope (maximum possible)	The maximum slope in a subsidence trough caused by working a critical area.
l	Station interval	The horizontal distance between two observation stations.
$\pm e$	Strain	Change in length per unit of length $e = \frac{\Delta l}{l}$ (Direction to be specified)
$\pm E$	Strain (maximum)	The maximum strain (lengthening or shortening) in a trough.
$\pm E_{max}$	Strain (maximum possible)	The maximum possible strain (lengthening or shortening) in a trough.
s	Subsidence	The vertical movement anywhere within a subsidence trough.
S	Subsidence (maximum)	The maximum vertical movement caused by working a sub-critical area.
S_{max}	Subsidence (maximum possible)	The maximum possible vertical movement of a point on the surface caused by working a critical area. (This corresponds, in general, to the seam thickness multiplied by the subsidence factor.)

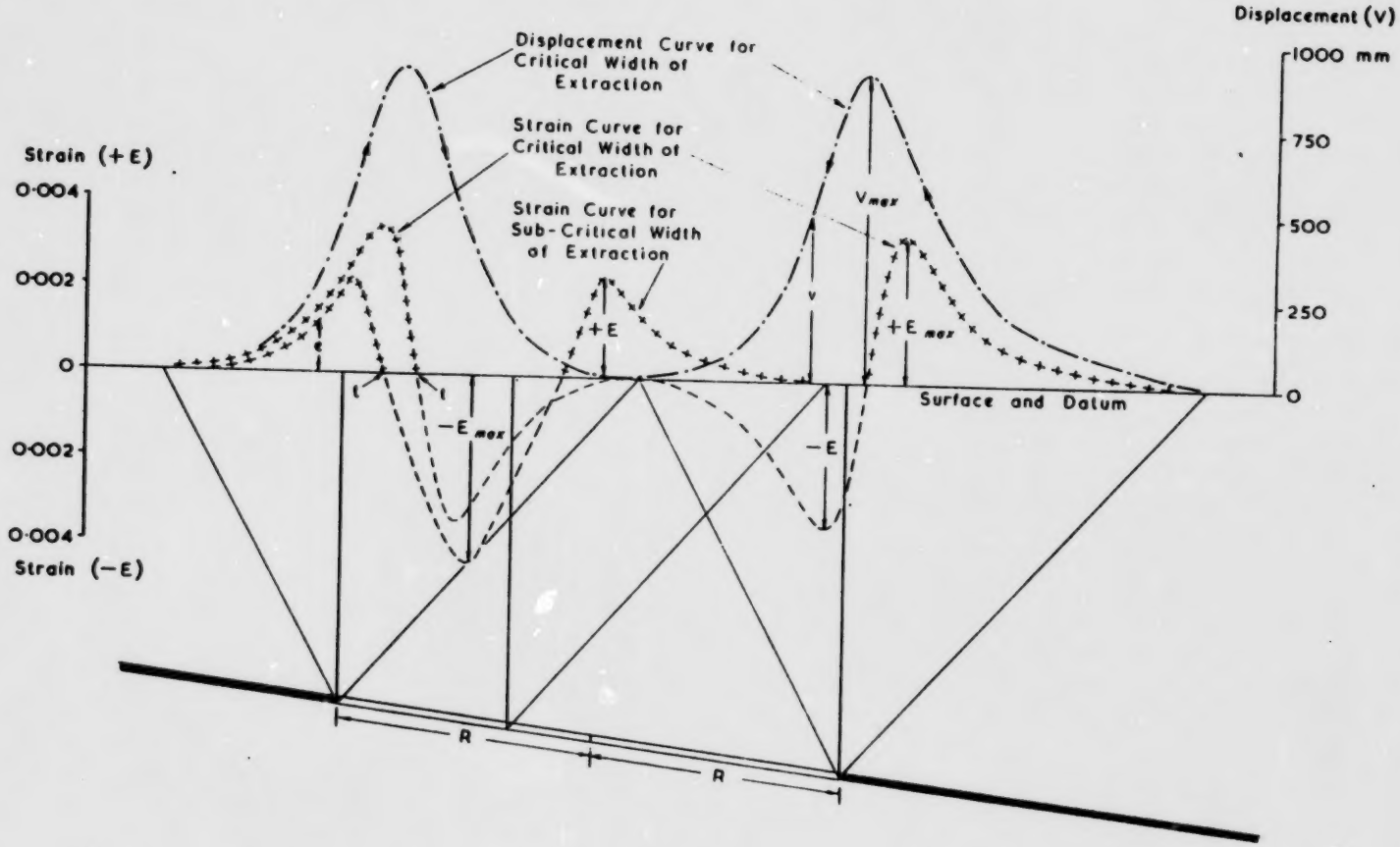


Fig. 2 Typical section through workings, illustrating standard symbols for strain and displacement.

Symbols	Terms	Definitions
$\frac{S}{2}$	Subsidence (half)	The point where subsidence has half the amplitude of that of the maximum in the profile. (This coincides with the transition point 't'.)
a	Subsidence factor	The ratio of complete subsidence to seam thickness, i.e. S_{max}/m.
	Subsidence development	The manner in which surface subsidence starts increases and finishes in relation to the position of the advancing face of an underground excavation.
	Subsidence development curve	A line on a graph depicting the changing subsidence of a surface point in relation to the position of the face of an excavation passing through the critical area of that point.
m	Thickness (of seam) extracted	
	Time-Subsidence curve	The values of subsidence of a given point on the surface plotted against time as a base.
t	Transition point	The point of transition between concave and convex curvature of a subsidence profile.
	Trough (subsidence)	The surface depression caused by subsidence.

Signs for Plans and Sections of Ground Movements

Name	Shown	Colour
Subsidence curve	———	Black
Displacement curve	· — · ◄ · — ·	Black
Lengthening curve	+ + + + + + + +	Black with red verge
Shortening curve	- - - - - - - - - - - -	Black with blue verge
Slope curve	– –/– –/– –/– –/– –	Black
Curvature curve	– –U– –U– –U– –U– –	Black
Lines of equal subsidence	——— 200 ———	Black
Displacement directions and values	10 = v	Black
Lengthening zone		Red verge
Shortening zone		Blue verge

APPENDIX 2

VARIOUS MODELS OF THE CLOSED FORM
ELASTIC SOLUTION
(From Berry (8))

APPENDIX

(i) Stress-strain relations

A layered medium will be transversely isotropic with respect to an axis of symmetry normal to the layers. If this is taken to be the y-axis, the stress-strain relations have the form—

$$\left.\begin{aligned} E_x\,\epsilon_x &= \sigma_x - \nu_y\,\sigma_y - \nu_z\,\sigma_z, \\ E_x\,\epsilon_y &= -\nu_y\,\sigma_x + (E_x/E_y)\sigma_y - \nu_y\,\sigma_z, \\ E_z\,\epsilon_z &= -\nu_z\,\sigma_x - \nu_y\,\sigma_y + \sigma_z, \\ 2M\gamma_{yz} &= \tau_{yz}, \\ E_z\gamma_{zx} &= (1+\nu_z)\,\tau_{zx}, \\ 2M\gamma_{xy} &= \tau_{xy}. \end{aligned}\right\} \quad \ldots \text{(A.1)}$$

The physical significance of the five constants is indicated in Fig. 4.

In a state of plane strain, such that there is no component of displacement in the z-direction $\epsilon_z = \gamma_{yz} = \gamma_{zx} = 0$ and the stress-strain relations reduce to the following three—

$$\begin{aligned} E_x\,\epsilon_x &= (1+\nu_z)\,[(1-\nu_z)\,\sigma_x - \nu_y\,\sigma_y], \\ E_x\,\epsilon_y &= (1+\nu_z)\,[-\nu_y\,\sigma_x + (1-\nu_z)\,\sigma_y/k_1^2], \\ 2M\gamma_{xy} &= \tau_{xy}, \end{aligned}$$

where k_1 is given by equation (1), while—

$$\sigma_z = \nu_z\,\sigma_x + \nu_y\,\sigma_y.$$

It will be seen that the plane strain relations can be expressed in terms of only four constants, namely—

$$\frac{1-\nu_z^2}{E_x}, \quad -\frac{\nu_y(1+\nu_z)}{E_x}, \quad \frac{1-\nu_z^2}{k_1^2 E_x}, \quad \frac{1}{2M}.$$

Berry and Sales (23) have referred to the third of these as s_{11}^{22}, and this symbol is used elsewhere in the present paper.

(ii) The critical constants

The constants k_1, k_2 defined by equation (1) are related to another pair of numbers, α_1 and α_2, by the equations—

$$\begin{aligned} \alpha_1 &= \left(\frac{k_1+k_2}{2}\right)^{\frac{1}{2}} + \left(\frac{k_2-k_1}{2}\right)^{\frac{1}{2}}, \\ \alpha_2 &= \left(\frac{k_1+k_2}{2}\right)^{\frac{1}{2}} - \left(\frac{k_2-k_1}{2}\right)^{\frac{1}{2}}. \end{aligned} \quad \ldots \text{(A.2)}$$

The constants are critical in the sense that the formula for any particular component of stress, strain or displacement takes different forms according to which of the following relations holds—

(a) $k_1 < k_2$ (α_1, α_2 real, unequal)
(b) $k_1 = k_2$ (α_1, α_2 real, equal)
(c) $k_1 > k_2$ (α_1, α_2 complex conjugates)

Although α_1 and α_2 are more fundamental in the mathematical analysis, it is generally more convenient to express relationships in terms of k_1 and k_2, as they are always real. However, the interest of the present work lies in case (a), and the formulae for the elastic field are then expressed more easily in terms of α_1 and α_2. When $k_1 = 2$ and $k_2 = 10$, the values of α_1 and α_2 are 4·4495 and 0·4495.

(iii) Two-dimensional (plane strain) formulae for $k_1 < k_2$

(a) Complete closure over $-a < x < a$ at depth h; surface subsidence and strain—

$$-\frac{v_0(x)}{t} = \frac{1}{\pi(\alpha_1-\alpha_2)}\left[\alpha_1 \tan^{-1}\frac{2ah_1}{x^2-a^2+h_1^2} - \right.$$

$$\left. - \alpha_2 \tan^{-1}\frac{2ah_2}{x^2-a^2+h_2^2}\right], \quad \ldots \text{(A.3)}$$

$$\frac{1}{t}\frac{du_0(x)}{dx} = \frac{2a\alpha_1\alpha_2}{\pi(\alpha_1-\alpha_2)}\left(\frac{X_1}{R_1^2} - \frac{X_2}{R_2^2}\right), \quad \ldots \text{(A.4)}$$

$h_j = h/\alpha_j, \quad X_j = x^2 - a^2 - h_j^2,$

$R_j = (X_j^2 + 4h_j^2x^2)^{\frac{1}{2}}, \quad j = 1, 2.$

(b) Non-closure over $-a < x < a$ at depth h; surface subsidence and strain—

$$-\frac{v_0(x)}{ps_{22}^{22}} = \frac{2^{\frac{1}{2}}(\alpha_1+\alpha_2)}{\alpha_1-\alpha_2} \times$$

$$\times\ [\alpha_1(R_1 - X_1)^{\frac{1}{2}} - \alpha_2(R_2 - X_2)^{\frac{1}{2}}], \quad \ldots \text{(A.5)}$$

$$\frac{1}{ps_{22}^{22}}\frac{du_0(x)}{dx} = \frac{2^{\frac{1}{2}}\alpha_1\alpha_2(\alpha_1+\alpha_2)x}{\alpha_1-\alpha_2} \times$$

$$\left[\frac{R_1 + X_1 + 2h_1^2}{R_1(R_1+X_1)^{\frac{1}{2}}} - \frac{R_2 + X_2 + 2h_2^2}{R_2(R_2+X_2)^{\frac{1}{2}}}\right], \quad \ldots \text{(A.6)}$$

(These are not exact non-closure formulae but hold when the convergence across the panel is given by—

$$2ps_{22}^{22}(\alpha_1+\alpha_2)(a^2-x^2)^{\frac{1}{2}},$$

which occurs when the free surface is too far away to exert any influence at the panel.)

(c) Partial closure over $-b < x < b$ of a panel occupying $-a < x < a$ of $y = 0$ in an infinite medium; stress components—

$$\sigma_y = -\frac{p}{\alpha_1-\alpha_2}\left\{\frac{1}{\alpha_1}\left[1 - \left(\frac{S_1+T_1}{2Y_1}\right)^{\frac{1}{2}}\right] - \right.$$

$$\left. - \frac{1}{\alpha_2}\left[1 - \left(\frac{S_2+T_2}{2Y_2}\right)^{\frac{1}{2}}\right]\right\}, \quad \ldots \text{(A.7)}$$

$$\sigma_y = \frac{p}{a_1 - a_2}\left\{a_1\left[1-\left(\frac{S_1+T_1}{2Y_1}\right)^{\frac{1}{2}}\right] - \right.$$

$$\left. -a_2\left[1-\left(\frac{S_2+T_2}{2Y_2}\right)^{\frac{1}{2}}\right]\right\}, \quad \ldots \text{(A.8)}$$

$$\tau_{xy} = -\frac{p}{a_1 - a_2}\left\{\left(\frac{S_1-T_1}{2Y_1}\right)^{\frac{1}{2}} - \right.$$

$$\left. -\left(\frac{S_2-T_2}{2Y_2}\right)^{\frac{1}{2}}\right\}\cdot\frac{xy}{|xy|}, \quad \ldots \text{(A.9)}$$

$$S_j = [T_j^2 + 4x^2y_j^2(a^2-b^2)^2]^{\frac{1}{2}},$$

$$T_j = (x^2 - y_j^2 - a^2)(x^2 - y_j^2 - b^2) + 4x^2y_j^2,$$

$$Y_j = (x^2 - y_j^2 - a^2)^2 + 4x^2y_j^2,$$

$$y_j = y/a_j, \qquad j = 1, 2.$$

(iv) Three-dimensional treatment

Complete closure over the semi-infinite strip, $-a < x < a$, $-\infty < z < 0$ at a depth h; surface subsidence—

$$-\frac{v_0(x,z)}{t} = -\frac{v_0(x)}{t} - \frac{1}{2\pi(a_1 - a_2)} \times$$

$$\times\left\{a_1\left[\tan^{-1}\frac{z(a+x)}{h_1A_1} + \tan^{-1}\frac{z(a-x)}{h_1B_1}\right] - \right.$$

$$\left. -a_2\left[\tan^{-1}\frac{z(a+x)}{h_2A_2} + \tan^{-1}\frac{z(a-x)}{h_2B_2}\right]\right\}, \quad \ldots \text{(A.10)}$$

where $A_j = [(a+x)^2 + z^2 + h_j^2]^{\frac{1}{2}}$

$B_j = [(a-x)^2 + z^2 + h_j^2]^{\frac{1}{2}}$

$h_j = h/a_j, \quad j = 1, 2,$

and $v_0(x)$ is given by (A.3), the two-dimensional formula for subsidence.

APPENDIX 3

VARIOUS MODELS OF THE FACE ELEMENT SOLUTION
(from Salamon (63))

HOMOGENEOUS ISOTROPIC MODEL

Infinite depth

If the face element is assumed to be in infinite depth, then the seam is a plane of symmetry, and no shear stress is exerted on it. Moreover, it is sufficient to consider only the half space for our solution. Placing the origin of the co-ordinate system in the seam, the boundary conditions are:

$$z = 0 \begin{cases} \tau_{xz} = \tau_{yz} = 0 & \\ w = -\frac{1}{2} s_z & \text{at } \Delta A \\ w = 0 & \text{outside } \Delta A \end{cases}$$

$$z = \infty \quad w = v = v = 0 \qquad \ldots\ldots\ldots(39)$$

the displacements, satisfying the equations of equilibrium

$$\frac{\partial e}{\partial x} - (1 - 2v)\,\nabla^2 u = 0$$

$$\frac{\partial e}{\partial y} + (1 - 2v)\,\nabla^2 v = 0$$

$$\frac{\partial e}{\partial z} + (1 - 2u)\,\nabla^2 w = 0 \ldots\ldots\ldots\ldots\ldots\ldots\ldots(40)$$

where $e = \frac{\partial u}{\partial x} + \frac{\partial v}{\partial y} + \frac{\partial w}{\partial z}$ and $\nabla^2 = \frac{\partial^2}{\partial x^2} + \frac{\partial^2}{\partial y^2} + \frac{\partial^2}{\partial z^2}$ $\ldots\ldots\ldots\ldots\ldots\ldots\ldots(41)$

and induced by either normal stresses or normal displacements on the boundary of the semi-infinite space are[6]:

$$w = 2(1 - \nu)\,\Psi - z\frac{\partial \Psi}{\partial z}$$

$$u = -\frac{\partial}{\partial x}\left[(1 - 2\nu)\varphi - z\Psi\right]$$

$$v = -\frac{\partial}{\partial y}\left[(1 - 2\nu)\varphi - z\Psi\right] \quad \ldots\ldots\ldots\ldots\ldots\ldots (42)$$

where $\Psi = \dfrac{\partial \varphi}{\partial z}$ ..... (43)

and $\nabla^2\varphi = 0$ ..... (44)

From eqn. 42 the boundary conditions can be written in terms of the harmonic function φ:

$$\left(\frac{\partial \varphi}{\partial z}\right)_{z=0} = -\frac{s_z}{4(1-\nu)} \quad \text{at } \Delta A$$

$$\left(\frac{\partial \varphi}{\partial z}\right)_{z=0} = 0 \quad \text{outside } \Delta A \quad \ldots\ldots\ldots\ldots\ldots\ldots (45)$$

These conditions indentify φ with the inverse potential:

$$\varphi = \frac{s_z\,\Delta A}{8\pi(1-\nu)}\,\frac{1}{(r^2 + z^2)^{1/2}} \quad \ldots\ldots\ldots\ldots\ldots\ldots (46)$$

the derivative of which, with respect to z, is zero everywhere except at the origin on $z = 0$, and it satisfies the Laplace equation (eqn. 44).

From this result, after performing the derivations prescribed by eqns. 43 and 42, taking into account the definitions of the displacement kernels (eqns. 8, 10, 11) functions F_{1a}, F_{2a}, F_{1m}, and F_{2m} can be deduced. The results are given by eqn. 22.

Finite depth

The superposition of the stress and displacement field of the actual and mirror image face elements results, at the plane of symmetry in the following stress components (Fig. 3):

$$z = 0 \quad \Delta\tau_{xza} - \Delta\tau_{xzm} = 0\,,\ \Delta\tau_{yza} - \Delta\tau_{yzm} = 0\,,\ \Delta\sigma_{za} = \Delta\sigma_{zm} \quad \ldots\ldots (47)$$

Since the surface ($z = 0$) is stress-free, the problem is reduced to finding the stress and displacement fields, in a half-space ($z \geq 0$), induced by the following boundary conditions:

$$z = 0 \quad \tau_{xzs} = \tau_{yzs} = 0 \quad \sigma_{zs} = -2(\Delta\sigma_{zm})_{z=H}$$

$$z = \infty \quad w_s = u_s = v_s = 0 \quad \ldots\ldots\ldots\ldots\ldots\ldots (48)$$

These boundary conditions allow the application of the fundamental results given by eqns. 42-44. From these formulae and the Hooke's law one can express the σ_z stress component in the following manner:

$$\sigma_z = \frac{E}{1-\nu}\left(\frac{\nu e}{1+2\nu} - \frac{\partial w}{\partial z}\right) = \frac{E}{1+\nu}\left(\frac{\partial \Psi}{\partial z} - z\frac{\partial^2 \Psi}{\partial z^2}\right) \quad \ldots\ldots(49)$$

This relationship permits the derivation of the displacement field due to the specifications of eqn. 48. The actual derivation is best performed with the aid of Hankel's integral transform, which is defined, for example, for function φ, as[7]:

$$\bar{\varphi} = \int_0^\infty r\,\varphi\,J_0(pr)\,dr \quad , \quad \varphi = \int_0^\infty p\,\bar{\varphi}\,J_0(pr)\,dp \quad \ldots\ldots(50)$$

Let us now introduce function Ψ_s representing the new displacement field. The boundary conditions of eqn. 48 can be expressed, in terms of the transforms of Ψ_s and $\Delta\sigma_{zm}$, in the following form (eqn. 49):

$$\left(\frac{\partial \overline{\Psi}_s}{\partial z}\right)_{z=0} = -\frac{2(1+\nu)}{E}\left(\overline{\Delta\sigma_{zm}}\right)_{z=H} \quad \ldots\ldots(51)$$

Since Ψ_s is a solution of the Laplace equation (eqns. 43 and 44) its transform, finite for large values of z, can be taken as:

$$\overline{\Psi}_s = c(p)\,e^{-pz} \quad \ldots\ldots(52)$$

Also, the transform of φ (eqn. 46)

$$\bar{\varphi} = \frac{s_z\,\Delta A}{8\pi(1-\nu)}\,\frac{1}{p}\,e^{-pz} \quad \ldots\ldots(53)$$

derived from the first definition of eqn. 50 by making use of some results quoted by Sneddon[8], yield, using eqns. 43 and 49, the transform of $\Delta\sigma_{zm}$,

$$\overline{\Delta\sigma_{zm}} = \frac{E\,s_z\,\Delta A}{8\pi(1-\nu^2)}\,p\,(1-pz)\,e^{-pz} \quad \ldots\ldots(54)$$

The boundary conditions (eqn. 51) with the expression for $\overline{\Delta\sigma_{zm}}$ define the unknown function (p) in Ψ_s (eqn. 52). Omitting the algebraic details $\overline{\Psi}_s$ is defined by the following integral:

$$\Psi_s = \frac{s_z\,\Delta A}{4\pi(1-\nu)}\int_0^\infty p(1+pH)\,e^{-p(z+H)}\,J_0(pr)\,dp \quad \ldots\ldots(55)$$

which, after evaluation[8], yields (see eqn. 23):

$$\Psi_s = \frac{s_z\,\Delta A}{4\pi(1-\nu)}\left[\frac{z}{(r^2+z_2^2)^{3/2}} - \frac{3\,H\,z_2}{(r^2+z_2^2)^{5/2}}\right] \quad \ldots\ldots(56)$$

The functions F_{1s} and F_{2s} resulting from this formula are given by eqns. 24 and 25 respectively.

HOMOGENEOUS TRANSVERSELY ISOTROPIC MODEL

The derivation is based on the fundamental results of Shield as given by Berry and Sales[3]. He has shown that a displacement in a half-infinite space, satisfying simultaneously the Hooke's law (eqn. 26) and the equations of equilibrium, and fulfilling the conditions $\tau_{xz} = \tau_{yz} = 0$ on $z = 0$, is obtained by putting:

$$w = \frac{q_1}{1 - q_1} \Psi(x, y, z_1^*) - \frac{q_2}{1 - q_2} \Psi(x, y, z_2^*)$$

$$u = \frac{\partial}{\partial x} \left[\frac{\alpha_1}{1 - q_1} \varphi(x, y, z_1^*) - \frac{\alpha_2}{1 - q_2} \varphi(x, y, z_2^*) \right]$$

$$v = \frac{\partial}{\partial y} \left[\frac{\alpha_1}{1 - q_1} \varphi(x, y, z_1^*) - \frac{\alpha_2}{1 - q_2} \varphi(x, y, z_2^*) \right] \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots (57)$$

provided the conditions, with regard to $\Psi(x, y, z)$ and $\varphi(x, y, z)$ imposed by eqns. 43, 44 are satisfied and

$$z_j^* = \frac{z}{\alpha_j} \qquad j = 1, 2 \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots (58)$$

where α_1 and α_2 are two roots of the equation

$$c_{11} c_{44} \alpha^4 - [c_{13}(2c_{44} + c_{13}) - c_{11} c_{33}] \alpha^2 + c_{33} c_{44} = 0 \quad \ldots\ldots\ldots\ldots (59)$$

with positive real parts. Also the constants q_1 and q_2 are calculated from the following expression:

$$q_j = \frac{c_{11}\alpha^2_j - c_{44}}{c_{13} + c_{44}} \qquad j = 1, 2 \quad \ldots\ldots\ldots\ldots\ldots\ldots (60)$$

In the latter two expressions the constants $c_{11} \ldots c_{44}$ are the elastic co-efficients occurring in the inverse of eqn. 26[3]. For subsequent use, the σ_z component can be expressed in the following manner:

$$\sigma_z = c_{44} \left[\alpha_1 \frac{\partial}{\partial z_1^*} \Psi(x, y, z_1^*) - \alpha_2 \frac{\partial}{\partial z_2^*} \Psi(x, y, z_2^*) \right] \quad \ldots\ldots\ldots\ldots\ldots (61)$$

Infinite depth

The method of solution used in the case of the isotropic model can also be applied here. The boundary conditions of eqn. 39 take the following form, on the basis of eqns. 43 and 57:

$$\left(\frac{\partial \varphi}{\partial z}\right)_{z=0} = -\frac{(1 - q_1)(1 - q_2) s_z}{2(q_1 - q_2)} \quad \text{at} \quad \Delta A$$

$$\left(\frac{\partial \varphi}{\partial z}\right)_{z=0} = 0 \quad \text{outside } \Delta A \quad \ldots\ldots\ldots (62)$$

which identifies $\varphi(x, y, z)$, apart from a constant multiplier, with eqn. 46 (see eqn. 45).

$$\varphi(x, y, z) = \frac{(1 - q_1)(1 - q_2) s_z \Delta A}{4\pi(q_1 - q_2)} \frac{1}{(r^2 - z^2)^{1/2}} \quad \ldots\ldots\ldots\ldots (63)$$

This result allows the derivation of the infinite depth displacement kernels on the basis of eqns. 43 and 57.

Finite depth

Using again Hankel's transform (eqn. 50) the new boundary conditions are (eqns. 48, 61):

$$\left(\frac{\partial \overline{\Psi}_s}{\partial z}\right)_{z=0} = -\frac{2}{c_{44}(\alpha_1 - \alpha_2)} \left(\overline{\Delta\sigma}_{zm}\right)_{z=H} \quad \ldots\ldots\ldots\ldots(64)$$

Since the transform of φ only differs from eqn. 53 by a known co-efficient (compare eqns. 46 and 63) the transform of $\overline{\Delta\sigma}_{zm}$ can be written from eqn. 61 as follows:

$$\overline{\Delta\sigma}_{zm} = c_{44} \frac{(1+q_1)(1+q_2)\, s_z\, \Delta A}{4\pi(q_1 - q_2)}\, p \left(\alpha_1 e^{-pz_1^*} - \alpha_2 e^{-pz_2^*}\right) \quad \ldots\ldots\ldots\ldots(65)$$

Taking the transform of Ψ_s again in the form given by eqn. 52, the boundary conditions (eqn. 64) together with the formula for $\overline{\Delta\sigma}_{zm}$ yield:

$$\Psi(x, y, z) = \frac{(1+q_1)(1+q_2)\, s_z\, \Delta A}{2\pi(q_1 - q_2)(\alpha_1 - \alpha_2)} \int_0^\infty p \left(\alpha_1 e^{-pH_1} - \alpha_2 e^{-pH_2}\right) e^{-pz} J_0(pr)\, dp \quad (66)$$

$$H_j = \frac{H}{\alpha_j} \qquad j = 1, 2$$

which after integration gives[8]:

$$\Psi_s(x, y, z) = \frac{(1-q_1)(1+q_2)\, s_z\, \Delta A}{2\pi(q_1 - q_2)(\alpha_1 - \alpha_2)} \left\{ \frac{\alpha_1 (z - H_1)}{[r^2 - (z - H_1)^2]^{3/2}} - \frac{\alpha_2 (z - H_2)}{[r^2 - (z - H_2)^2]^{3/2}} \right\} \quad (67)$$

Results given by eqns. 63 and 67 completely define the face element solution for the transversely isotropic model. A considerable amount of algebraic work is required, however, to obtain the solution in practical form. The reason for this is that α_1 and α_2 are either positive quantities or complex conjugates.

MULTI-MEMBRANE MODEL

In this simple model the boundary conditions can only involve one displacement (w) or one stress (σ_z) component. Also, it will be noticed that the general equation of this model (eqn. 36) can be written in the following form:

$$\frac{\partial^2 w}{\partial x^2} + \frac{\partial^2 w}{\partial y^2} + \frac{\partial^2 w}{\partial \left(\frac{z}{a}\right)^2} = 0 \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots (68)$$

that is to say w satisfies the Laplace equation in the x, y, $\frac{z}{a}$ co-ordinate system.

Infinite depth

The boundary conditions of the face element in this case are:

$$z = 0 \begin{cases} w = -\frac{1}{2} s_z & \text{at} \quad \Delta A \\ w = 0 & \text{outside } \Delta A \end{cases}$$

$$z = \infty \quad w = 0 \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots (69)$$

If one compares these conditions with those prescribed by eqn. 45 and remembers eqns. 43, 44, then it is apparent w must be identical, apart from a constant multiplier, to Ψ (see eqns. 43 and 46), after replacing z by $\frac{z}{a}$.

$$\Delta w = -\frac{s_z \, \Delta A}{4\pi} \frac{\left(\frac{z}{a}\right)}{\left[r^2 + \left(\frac{z}{a}\right)^2\right]^{3/2}} \qquad \ldots\ldots\ldots\ldots\ldots (70)$$

Finite depth

If the displacement field due to a diverging mirror image face element (Fig. 3) is superimposed on the above solution, then the plane half way between them will be free of normal stress. Since the normal stress is the only stream component which may occur in the boundary conditions, the above mentioned two displacement fields represent the complete solution in this model. The expression of superposition is given by eqn. 31 and the displacement kernel by eqn. 38 as deduced from eqn. 70.

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APPENDIX 4

FINITE ELEMENT REFERENCES
(From Turcke (83))

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APPENDIX 5

SUBSIDENCE PREDICTION METHOD SPREADSHEETS

METHOD	EMPIRICAL	EMPIRICAL	EMPIRICAL
TYPE	GRAPHICAL	GRAPHICAL	MATHEMATICAL
NAME	N.C.B.	PILLAR	PROFILE FUNCTION
INPUT REQUIRED	- panel width - panel depth - seam thickness - panel advance - panel length - bed inclination - face advance rate	- pillar width - panel depth - seam thickness - extraction ratio	- max. subsidence - critical radius - distance to inflection pt. - K field constant.
TIME FACTOR	limited	no	no
PILLAR FACTOR	limited	yes	limited
COMPLEX GEOMETRY	no	limited	no
OUTPUT	- max. subsidence - subsidence profile - lateral strain - duration of subsidence	- max. subsidence	-subsidence profile

METHOD	EMPIRICAL	EMPIRICAL	EMPIRICAL
TYPE	MATHEMATICAL	MATHEMATICAL	MATHEMATICAL
NAME	INFLUENCE FUNCTION	COMPLEMENTARY INFLUENCE FUNCTION	INCREMENTAL DESIGN
INPUT REQUIRED	- max. subsidence - critical radius - seam thickness - mined mat'l influence function	- max. subsidence - critical radii - seam thickness - overburden inertia fn. - unmined mat'l crushing fn. - mined mat'l influence fn. - unmined mat'l influence fn.	- max. subsidence - seam thickness - panel length - panel width - panel depth
TIME FACTOR	yes	yes	no
PILLAR FACTOR	limited	- yes	limited
COMPLEX GEOMETRY	yes	yes	limited
OUTPUT	- subsidence profile	- subsidence profile	- max. subsidence - limited profile

METHOD	ANALYTICAL	ANALYTICAL	ANALYTICAL	ANALYTICAL
TYPE	CLOSED FORM	GENERALIZED INFLUENCE	NUMERICAL	NUMERICAL
NAME	ELASTIC TRANSVERSELY ISO-TROPIC MEDIUM	FACE ELEMENT	VOID VOLUME	FINITE ELEMENT
INPUT REQUIRED	- panel width - panel depth - vertical roof convergence - elastic parameters of rock (E,u,G)	- panel depth - vertical roof convergence - elastic parameters of rock (E,u,G)* *depending on model	- panel depth - seam thickness - time-independent bulking factor - time-dependent bulking factor (both factors require knowledge of stress conditions)	- elastic and time dependent parameters of rock* - discontinuous properties and geometries - stress and boundary conditions *depending on model
TIME FACTOR	no	no	yes	yes*
PILLAR FACTOR	no	limited	yes	yes
COMPLEX GEOMETRY	no	yes	yes	yes
OUTPUT	- max. subsidence - subsidence profile - lateral strains - lateral stress	- max. subsidence - subsidence profile - lateral strains - lateral stress	- max. subsidence - subsidence profile - lateral strains - lateral stress	- max. subsidence - subsidence profile - lateral strains - lateral stress

APPENDIX 6

SUBSID PROGRAM ANALYSIS

TECHNICAL MEMORANDUM

Use of SUBSID-B to Predict Surface Subsidence Resulting from Potash Mining

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Appendix 2: Direct Calculation of Lambda
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1.0 INTRODUCTION

One of the major concerns with the undergound mining of potash is the prediction of surface subsidence resulting from extraction of the mineral beneath. Surface subsidence prediction is most easily facilitated through some form of numerical modelling such as finite element analysis. Due to the nature of potash, numerical models used in these circumstances must account for this mineral's mechanical non-linearities. Specifically, the stress-strain curve for potash is markedly non-linear and time-dependent when compared to the stress-strain curve for other rocks and minerals such as granite or limestone. Since the relationship between stress and strain is not linear, potash will exhibit varying increments of strain for constant increments of stress.

The Mining Technology Division purchased a non-linear finite element program in order to evaluate its effectiveness as a prediction tool. This program, called SUBSID-B, is based on a linearly or non-linearly behaving beam resting on a rigid foundation which may or may not be overlain by a linear elastic overburden. The original program, called SUBSID, was developed at the University of California for the USBM for the purpose of surface subsidence prediction resulting from the retorting of oil shales (10).

SUBSID has been successfully used to predict surface subsidence due to mining in both oil shale and coal mining. While this program appears to be particularly well suited to single seam potash mining, such an application was not found in the available literature. Consequently, the numerical input to the SUBSID-B model had to be first calibrated to reflect the mechanical properties of potash and its associated overburden. In order to be of predictive value, the input values were required to produce results similar to measured subsidence over existing potash workings.

Numerical input data were determined from published values of mechanical properties of potash and its associated overburden. Initial investigations indicated that the required mechanical data was somewhat scant. However, similar subsidence studies were much more abundant in the field of coal mining. Therefore, in order to substantiate the available potash data, mechanical properties used in the coal mining subsidence models were considered where applicable.

2.0 FINITE ELEMENT MESH CONSTRUCTION

The finite element method requires that the situation to be modelled be discretized or broken up into discrete elements. Because of SUBSID's simplifying assumptions regarding the problem geometry, generation of the mesh is quite simple although somewhat restrictive. The only choice available to the user is the selection of the element's "x" width. This parameter usually corresponds to the width of rooms and pillars.

Due to the "chevron" mine layout of CCP, a "reasonable" cross section could not be drawn to model an exact mine situation. However, the room and pillar layout which was modelled was geometrically exact. The exact dimensions of pillars and rooms were modelled and combined to produce the current mining extraction at CCP.

2.1 Model Size

With most finite element programs the overall size of the model is usually about eight (8) times the size of the maximum dimension being modelled. That is to say that if a 10 ft. square opening is to be modelled, the overall size of the finite element mesh should be about 80 ft. x 80 ft. centred about the square opening. Based largely on this rule-of-thumb, the first meshes were drawn. It was discovered that the amount of subsidence calculated by the model was largely dependent upon the length of "abutment pillar" separating the mine workings from the furthest boundary of the finite element mesh. Through trial-and-error, it was determined that this "size effect" gradually diminished once the overall length of the model was greater than twelve (12) times the length of the total mining area being modelled. The size effect was found to be eliminated when the total length of the finite element mesh was fifteen (15) times greater than the total length of the mining area being modelled.

This ratio corresponds well with the published data in the original SUBSID user's manual (10). The oil shale retorting test case in this reference uses an overall model length approximately thirteen (13) times the total length of the mining area which was modelled. Therefore, for the potash mining geometries being considered, the overall mesh length should be about fifteen (15) times greater than the total length of the mining area being modelled. In general, the greater this ratio, the lesser the size effects on the final results.

2.2 Boundary Conditions

While any node of the finite element mesh may be fixed with respect to displacement in the x (horizontal) direction, y (vertical) direction and θ (rotational) direction, all nodes are free to move in any of these directions in this potash mining

model. Fixing a node with respect to displacement in any of these directions is usually done to simulate a boundary condition. There are only two (2) boundaries which may be fixed in this manner. These boundaries may be referred to as the left and right boundaries. The other two boundaries, top and bottom, may not be fixed in any manner. Firstly, the bottom boundary is already assumed to be rigidly fixed to a rigid foundation. The nodes along the bottom boundary may not displace in any manner. Secondly, the top boundary, which represents the surface, may not be fixed in any manner, otherwise, the true surface displacement would not be correctly calculated.

In order to reduce the computational time required for each program run, symmetry about the y axis is used. A boundary condition along this axis of symmetry is required. In "real life", this boundary would be abutted against the other half of the model. Hence, displacement in the x direction would be impossible although displacement in the y and θ directions would be possible. This would be analogous to placing "rollers" along this boundary.

The external boundary or the boundary furthest away from the mine workings (i.e. the right boundary) should be unrestricted in displacement according to the user documentation of the original program(10). The results of the program run will usually indicate excessive surface displacements near this unrestricted boundary. However, this is normal and should be disregarded.

A diagram of the final mesh which was used can be found in Figure 1.

3.0 MATERIAL PROPERTIES SELECTION

The mechanical properties of the materials which were modelled were not known prior to any of the program runs. The following data, extracted from various sources, represent mechanical properties obtained largely through laboratory testing. The available values were found to be expressed in a wide variety of units. All units have been converted to pounds-feet (to match existing mining geometrics) and SI (to facilitate ease of future reference). Appendix 1 contains the conversions which were required.

3.1 Overburden Properties

3.1.1 Young's Modulus

Several values of Young's Modulus were found for overburden of both coal and potash mines. These values are summarized in Table 1. The values range from 2.160×10^6 psf. to 432.00×10^6 psf. The average value of 95.00×10^6 psf was used. This value was close to overburden values corresponding to shale, sandstone and other non-categorized coal overburdens.

3.1.2 Poisson's Ratio

Poisson's Ratio was determined from the same sources as was Young's Modulus. These values are summarized in Table 2. The values ranged from 0.14 to 0.25. The average value of 0.22 was used.

3.1.3 Weight Density

Many of the same references also provided weight density information. This information is summarized in Table 3. Overburden weight densities varied from 130.5 lbs/ft^3 to 171.9 lbs/ft^3. The average value of 150 lbs/ft^3 was used. This value compared favourably with weight densities of sandstone, siltstone and other non-characterised coal overburdens.

3.2 Potash Properties

3.2.1 Compressive Strength

The compressive strength of potash was fairly well documented from a variety of sources. Specific values for anhydrite, rock salt, hard potash salt and carnalite were found. These values are summarized in Table 4. The selected value of Co was 1025.00 $\times 10^3$ lbs/ft^2. This was the compressive strength for hard potash salt reported by Hofer and Menzel (5). Considering the range of values reported, the selected value is also approximately equal to the average value.

3.2.2 Weight Density

Relatively few references were found which specified the weight density of potash. The available values are summarized in Table 5. A weight density of 136 lbs/ft^3 was used which corresponded to the quoted value for rock salt(2). This value correlated well with the average value of Table 5.

3.2.3 Poisson's Ratio

Perhaps surprisingly, many references were found quoting values of Poisson's Ratio for potash and other salts. These values are summarized in Table 6. Values ranged from 0.25 to 0.35. One reference stated that Poisson's Ratio for most salts is about 0.35 (13). However, using the data at hand, an average value of 0.30 was used.

3.2.4 Porosity

SUBSID uses porosity to determine the initial strains produced under low stresses. It assumes that these first strains are produced by a decrease in pore volume and micro-crack closing. Only one reference was found to discuss the porosity of potash (1). In general, it stated that a porosity of 5 to 10% could be expected under 6 to 12 m of salt and then gradually decrease to 0% with deeper coverings. Since the potash seam being modelled lies about 3300 ft below surface, it was felt that the porosity of potash at this elevation would be about 0% mainly due to the superincumbant load. In any event, the final surface subsidence due to pore volume decrease in a 10.5 ft. high pillar would be very much smaller than subsidence due to other pillar deformations and roof-to-floor convergence.

3.2.5 Power Strain

The power strain is used to describe the shape of the stress-strain curve in the region of initial pore volume decrease. A value of two (2) was used because it is largely considered to be a constant by the available literature. However, any value could have been used in this case and the identical results would have been obtained. Since the porosity was set to 0%, strain due to pore volume decrease is not possible and the shape parameter (i.e. power strain) becomes meaningless in this instance.

3.2.6 Lambda

The lambda parameter is the counterpart to the power-strain parameter whereby lambda describes the shape of the stress-strain curve after total pore volume decrease has been achieved. Very little documentation was available. The lambda value was selected on a trial-and-error basis. Having already selected all other mechanical parameters and knowing the measured subsidence,

successive runs of the SUBSID model were performed until the proper amount of subsidence was achieved. This resulted in a final value of 10 being used.

This value is substantiated on an order-of-magnitude basis. Appendix 2 contains a manual calculation of lambda from potash stress-strain data from reference 1. This manual calculation yields a lambda of about 30. Other published values of lambda are 100 and 246 for oil shale and coal respectively. Hence, the manually calculated value for lambda substantiates the calibration value of lambda at least on an order-of- magnitude basis.

4.0 CONCLUSIONS

The previously discussed input data were entered into the SUBSID program. The program output can be found in Appendix 3. The program produces the desired results and has a high degree of correlation with other numerical models and calculation methods.

In general, the following items should be considered when running a SUBSID model:

1) The overall length of the model should be about 15 times the length of the total mining area being modelled to eliminate "end effects".

2) All nodes should be free to displace in either the x, y and θ directions. When symmetry is used, the axis of symmetry should be fixed so as not to allow displacement in the x direction along this axis. The external boundary should be free to displace in all directions.

3) The program usually indicates excessive surface displacement at the external boundary. This is normal and should be disregarded.

The following mechanical properties of potash and its overburden were used to produce the desired results:

Young's Modulus of Overburden	=	95.00×10^6 psf.
Poisson's Ratio of Overburden	=	0.22
Weight Density of Overburden	=	150 lbs/ft^3
Compressive Strength of Potash	=	1025.00×10^3 psf.
Weight Density of Potash	=	136 lbs/ft^3
Poisson's Ratio of Potash	=	0.30
Porosity of Potash	=	0
Lambda	=	10

The sensitivity of the calculated surface subsidence to the input values used for the elastic parameters was examined for this calibrated base case. The parameters examined included Young's Modulus of Overburden, Compressive Strength of Potash and Lambda. Poisson's Ratio was not considered because it is a second order elastic parameter. Weight density was not considered because its only effect is to determine the superincumbant load and it is felt that that the weight density parameters are known to a relatively high degree of accuracy. As previously stated, porosity and, hence, the power strain value do not enter into this model since porosity is about 0%.

The results of these sensitivity program runs are illustrated in Figure 2. It can be seen that the more vertically oriented line of Young's Modulus of overburden indicates that this parameter does not greatly influence the calculated surface subsidence (i.e. a large change in Young's Modulus of Overburden does not result in a correspondingly large change in calculated surface subsidence). Clearly the potash parameters of compressive strength and lambda exhibit a much greater influence on the calculated surface subsidence.

Interestingly, both of these parameters are contained in the same constitutive equation. After simplifying for porosity and power strain the equation becomes:

$$\sigma = Co(1-e^{-\lambda\varepsilon})\ldots \text{ See Appendix 2}$$

Therefore, it may be stated that the calibrated base case model is sensitive to the shape of the stress-strain curve for potash. However, Table 4 (Compressive Strength of Potash) would indicate that a suffficient amount of data is available for the compressive strength of potash at least for an upper/lower bounds type of analysis. Most of the uncertainty must then be contained in the lambda parameter. Therefore, further laboratory testing is required to more accurately determine the compressive strength of potash and in particular to determine a value for lambda. This could be done through detailed stress-strain testing.

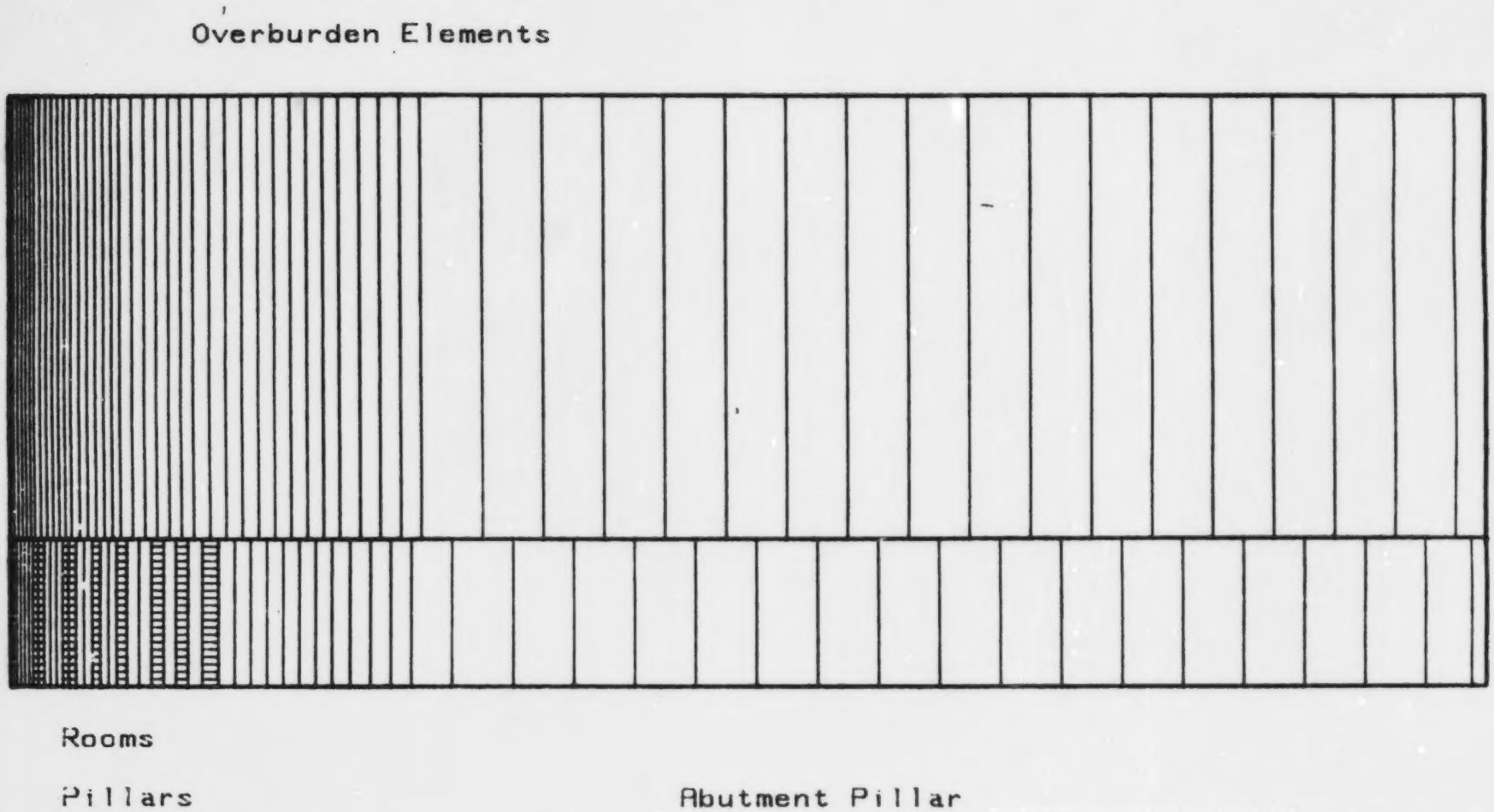


FIGURE 1: SUBSID Finite Element Mesh

(Disproportionate Horizontal And Vertical Scale)

TABLE 1 - Young's Modulus of Overburden

REF.	MATERIAL	Literature Units	Pounds/ft^2	GPa
8	Coal Overburden	15,000 psi	2.16×10^6	0.10
14	Slightly Indurated Sandstone	150 MPa	3.13×10^6	0.15
14	Coal Overburden (Siltstone)	180 MPa	3.76×10^6	0.18
14	Moderately Indurated Sandstone	200 MPa	4.18×10^6	0.20
3	Overburden Over Gob	5×10^4psi	7.20×10^6	0.34
8	Coal Overburden	414 MPa	8.65×10^6	0.41
6	Wet Overburden Over Coal	1.0×10^5 psi	14.40×10^6	0.69
3	Coal Overburden	20×10^4 psi	28.80×10^6	1.38
6	Coal Overburden	3×10^5 psi	43.20×10^6	2.07
10	Oil Shale Overburden	3000 MPa	62.67×10^6	3.00
8	Coal Overburden	4000 MPa	83.56×10^6	4.00
7	Coal Overburden	0.6×10^6 psi	86.40×10^6	4.14
12	Coal Overburden (Shale)	4.5×10^3 MPa	95.01×10^6	4.50
12	Coal Overburden (Sandstone)	7.22×10^3 MPa	150.80×10^6	7.22
12	Coal Overburden (Siltstone)	9.37×10^3 MPa	195.70×10^6	9.37
5	Sandstone Over Potash	130×10^3 kg/cm^2	266.30×10^6	12.75
7	Coal Overburden	2.0×10^6 psi	288.00×10^6	13.79
7	Coal Overburden	3.0×10^6 psi	432.00×10^6	20.69

Minimum = 2.16×10^6 lbs/ft^2 (0.10 GPa)
Maximum = 432.00×10^6 lbs/ft^2 (20.69 GPa)
Average = 98.62×10^6 lbs/ft^2 (4.72 GPa)
Selected = 95.00×10^6 lbs/ft^2 (4.55 GPa)

TABLE 2 - Poisson's Ratio of Overburden

Ref.	Material	Poisson's Ratio
12	Shale	0.14
12	Siltstone	0.22
12	Sandstone	0.22
10	Oil Shale Overburden	0.25
5	Potash Overburden	0.25
3	Overburden Over Gob	0.25

Minimum = 0.14
Maximum = 0.25
Average = 0.22
Selected = 0.22

TABLE 3 - Weight Density of Overburden

REF.	Material	Literature Units	Pounds/ft^3	KN/m^3
14	Slightly Indurated Sandstone	2090 kg/m^3	130.5	20.50
2	Sandstone	134 lbs/ft^3	134.0	21.05
14	Moderately Indurated Sandstone	2300 kg/m^3	143.5	22.54
12	Coal Overburden (Shale)	2305.7 kg/m^3	144.0	22.62
12	Coal Overburden (Siltstone)	2305.7 kg/m^3	144.0	22.62
12	Coal Overburden (Sandstone)	2305.7 kg/m^3	144.0	22.62
2	Sandstone	147 lbs/ft^3	147.0	23.09
14	Coal Overburden (Siltstone)	2360 kg/m^3	147.4	23.15
8	Coal Overburden	0.087 lbs/in^3	150.3	23.61
8	Coal Overburden	0.024 MN/m^3	152.8	24.00
5	Potash Overburden	2.5x10^3 kg/m^2	156.1	24.53
7	Coal Overburden	170 lbs/ft^3	170.0	26.71
10	Oil Shale Overburden	0.027 MPa/m	171.9	27.00

Minimum = 130.5 lbs./ft^3 (20.50 kN/m^3)

Maximum = 171.9 lbs/ft^3 (27.00 kN/m^3)

Average = 148.89 lbs/ft^3 (23.39 kN/m^3)

Selected = 150.00 lbs/ft^3 (23.57 kN/m^3)

TABLE 4 - Compressive Strength of Potash

REF	MATERIAL	Literature Units	Pounds/ft^2	MPa	
10	Oil Shale	3 MPa	62.67×10^3	3.00	
1	Carnalite	41 kgf/cm^2	84.05×10^3	4.02	*
1	Carnalite	62 kgf/cm^2	127.10×10^3	6.08	*
1	Carnalite	65 kgf/cm^2	133.30×10^3	6.38	*
1	Carnalite	66 kgf/cm^2	135.30×10^3	6.48	*
6	Lignite Coal	1400 psi	201.60×10^3	9.65	
5	Carnalite	100 kg/cm^2	204.90×10^3	9.81	*
1	Carnalitic Rock /Salt	130 kgf/cm^2	266.50×10^3	12.75	*
14	Coal	17.9 MPa	373.90×10^3	17.90	
8	Coal	21 MPa	438.70×10^3	21.00	
1	Rock Salt	400 kgf/cm^2	820.00×10^3	39.24	*
5	Rock Salt	500 kg/cm^2	1025.00×10^3	49.05	*
5	Hard Potash Salt	500 kg/cm^2	1025.00×10^3	49.05	*
5	Anhydrite	1200 kg/cm^2	2459.00×10^3	117.70	*

* Considered In Parameter Selection:

Average = 628.02×10^3 lbs/ft^2 (30.06 MPa)

Minimum = 84.05×10^3 lbs/ft^2 (4.02 MPa)

Maximum = 2459.00×10^3 lbs/ft^2 (117.70 MPa)

Selection = 1025.00×10^3 lbs/ft^2 (49.05 MPa)

TABLE 5 - Weight Density of Potash

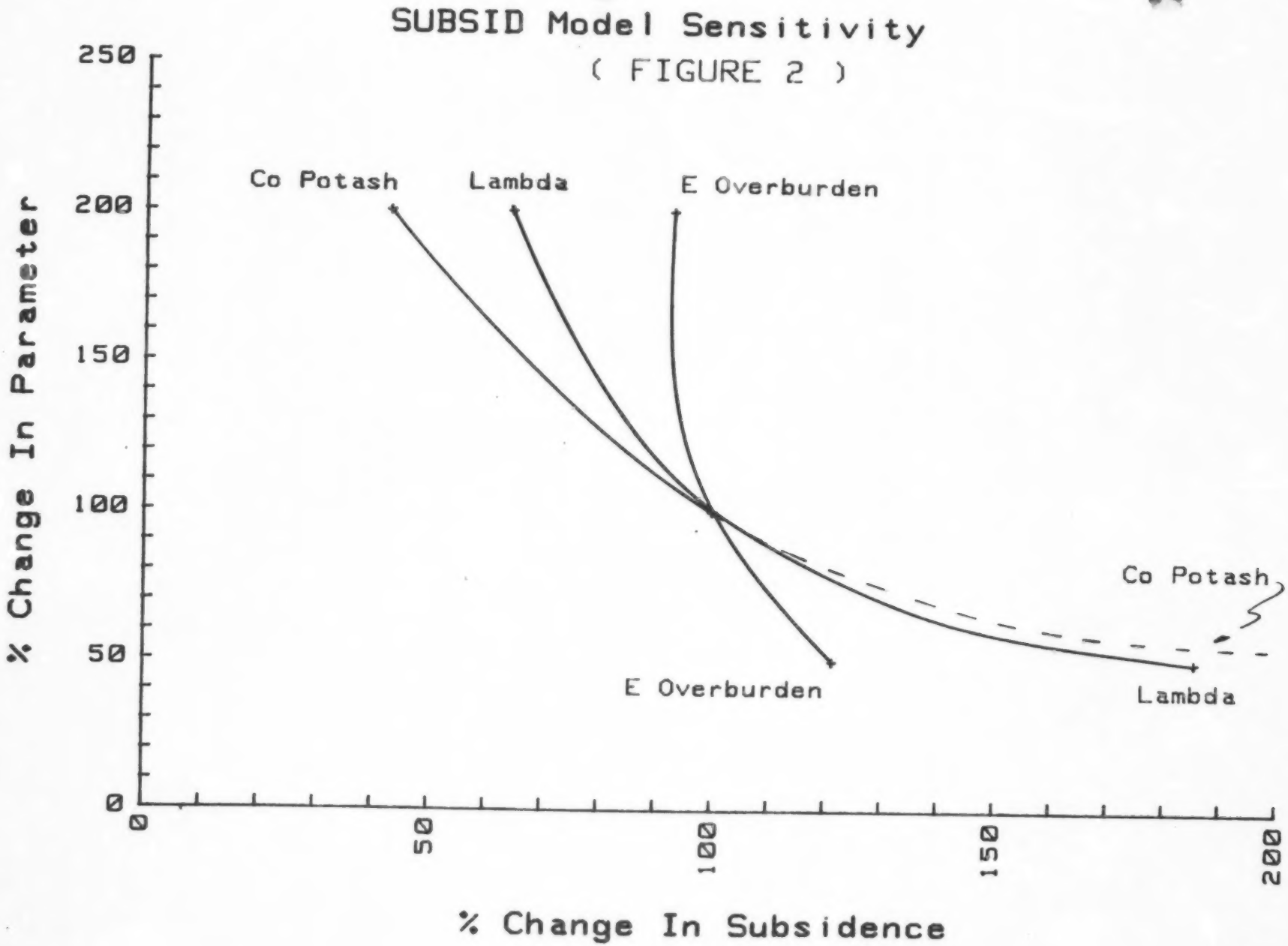
REF	MATERIAL	LITERATURE UNITS	POUNDS/FT^3	KN/M^3
2	Carnalite	S.G. = 1.602	99.96	15.70
2	Sylvanite	S.G. = 1.99	124.20	19.51
2	Rock Salt	136 lbs./ft^3	136.00	21.36
2	Gypsum	145 lbs./ft^3	145.00	22.78
4	Potash	S.G. = 2.5	156.00	24.51
2	Anhydrite	S.G. = 2.98	186.00	29.22

Minimum = 99.96 lbs/ft^3 (15.70 KN/m^3)
Maximum = 186.00 lbs/ft^3 (29.22 KN/m^3)
Average = 141.19 lbs/ft^3 (22.18 KN/m^3)
Selected = 136 lbs/ft^3 (21.36 KN/m^3)

TABLE 6 - Poisson's Ratio of Potash

REF	Material	Poisson's Ratio
5	Potash	0.25
11	Salt	0.27
11	Salt	0.31
9	Salt	0.33
13	Salt	0.35

Minimum = 0.25
Maximum = 0.35
Average = 0.30
Selected = 0.30


SUBSID Model Sensitivity
(FIGURE 2)
% Change In Parameter
250
200
150
100
50
0
Co Potash
Lambda
E Overburden
Co Potash
E Overburden
Lambda
0
50
100
150
200
% Change In Subsidence

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APPENDIX 7

UNIT CONVERSIONS

Unit Conversions from Literature Units

Conversions Used: $2.54 \frac{cm}{in}$

$0.224808 \frac{lbs}{N}$

$3.2808399 \frac{ft}{M}$

$9.81 \frac{N}{kgf}$

Physical Constants Used: $g = 9.81 \frac{m}{s^2}$

ρ water $= 62.4 \frac{lbs}{ft^3}$

ρ water $= 9.81 \times 10^3 \frac{N}{m^3}$

Conversions to Pounds - Feet

$$1 \frac{lbs}{in^2} = (1 \frac{lbs}{in^2})(12 \frac{in}{ft})^2 = 144 \frac{lbs}{ft^2}$$

$$1 \text{ Pa} = (1 \frac{N}{m^2})(\frac{m}{3.2808399 \text{ ft}})^2 (0.224808 \frac{lb}{N}) = 20.89 \times 10^{-3} \frac{lbs}{ft^2}$$

$$1 \frac{kg}{cm^2} = (1 \frac{kg}{cm^2})(100 \frac{cm}{m})^2 (9.81 \frac{m}{s^2})(\frac{m}{3.2808399 \text{ ft}})^2 (0.224808 \frac{lb}{N}) = 2.049 \times 10^3 \frac{lbs}{ft^2}$$

$$1 \frac{kg}{m^3} = (1 \frac{kg}{m^3})(9.81 \frac{m}{s^2})(\frac{m}{3.2808399 \text{ ft}})^3 (0.224808 \frac{lb}{N}) = 62.45 \times 10^{-3} \frac{lbs}{ft^3}$$

$$1 \frac{N}{m^3} = (1 \frac{N}{m^3})(0.244808 \frac{lb}{N})(\frac{m}{3.2808399 \text{ ft}})^3 = 6.366 \times 10^{-3} \frac{lb}{ft^3}$$

$$1 \frac{lb}{in^3} = (1 \frac{lb}{in^3})(\frac{12 \text{ in}}{ft})^3 = 1.728 \times 10^3 \frac{lb}{ft^3}$$

$$1 \frac{kgf}{cm^2} = (1 \frac{kgf}{cm^2})(9.81 \frac{N}{kgf})(0.224808 \frac{lb}{N})(\frac{2.54 \text{ cm}}{in})^2 (\frac{12 \text{ in}}{ft})^2 = 2.049 \times 10^3 \frac{lb}{ft^2}$$

Conversion to Meters-kilogram-Second (MkS)

$$1\,\frac{lb}{in^2} = (1\,\frac{lb}{in^2})(\frac{in}{2.54\ cm})^2(100\,\frac{cm}{m})^2\,(\frac{N}{0.224808\ lb}) = 6.895 \times 10^3\ Pa$$

$$1\,\frac{kg}{cm^2} = (1\,\frac{kg}{cm^2})(9.81\,\frac{m}{s^2})(100\,\frac{cm}{m})^2 = 98.10 \times 10^3\ Pa$$

$$1\,\frac{kg}{m^3} = (1\,\frac{kg}{m^3})(9.81\,\frac{m}{s^2}) = 9.81\,\frac{N}{m^3}$$

$$1\,\frac{lb}{ft^3} = (1\,\frac{lb}{ft^3})(3.2808399\,\frac{ft}{m})^3\,(\frac{N}{0.224808\ lb}) = 157.1\,\frac{N}{m^3}$$

$$1\,\frac{lb}{in^3} = (1\,\frac{lb}{in^3})(12\,\frac{in}{ft})^3(3.2808399\,\frac{ft}{m})^3\,(\frac{N}{0.224808\ lb}) = 271.4 \times 10^3\,\frac{N}{m^3}$$

$$1\,\frac{kgf}{cm^2} = (1\,\frac{kgf}{cm^2})(\frac{100\ cm}{m})^2(9.81\,\frac{N}{kgf}) = 98.10 \times 10^3\ Pa$$

APPENDIX 8

MANUAL CALCULATION OF LAMBDA

Direct Calculation of Lambda (λ)

$\sigma = (Co-k)(1-e^{-\lambda(\varepsilon-\varepsilon^*)}) + k$ (From Reference 10)

$Co = 60$ kgf/cm² (From Reference 1)

$\varepsilon^* = 0$ (From Reference 1)

$k = \frac{\lambda Co \varepsilon^*}{n + \lambda \varepsilon^*} = 0$ (Since $\varepsilon^* = 0$)

Therefore $\sigma = 60(1-e^{-\lambda\varepsilon})$

Using Measured stress-strain data (from Reference 1 Pg. 69) try different values of lambda (λ) until a "best fit" is found.

Measured Strain	Measured Stress	Try λ = 10 δ	Error	Try λ = 100 δ	Error	Try λ = 30 δ	Error
0.000	0	0.000	0.000	0.000	0.000	0.000	0.000
0.005	8	2.926	-5.074	23.608	15.608	8.358	0.358
0.010	14	5.710	-8.290	37.927	23.927	15.551	1.551
0.015	18	8.358	-9.642	46.612	28.612	21.742	3.742
0.020	23	10.876	-12.124	51.880	28.880	27.071	4.071
0.025	28	13.272	-14.728	55.075	27.075	31.658	3.658
0.030	33	15.551	-17.449	57.013	24.013	35.606	2.606
0.036	38	18.139	-19.861	58.361	20.361	39.624	1.624
0.040	42	19.781	-22.219	58.901	16.901	41.928	-0.072
0.047	48	22.500	-25.500	59.454	11.454	45.351	-2.649
0.050	51	23.608	-27.392	59.596	1.169	46.612	-4.388
0.058	58	26.406	-31.594	59.818	1.818	49.469	-8.531
0.060	60	27.071	-32.929	59.851	-0.149	50.082	-9.918
		ΣError = -226.802		ΣError = 199.669		ΣError = -7.948	
		Avg. Error Per Measurement = -17.446		Avg. Error per Measurement = 15.359		Avg. Error per Measurement = -0.611	

It can be seen that $\lambda \approx 30$ (perhaps slightly greater)

APPENDIX 9

COMPUTER OUTPUT

CCP SUBSIDENCE SIMULATION USING LITERATURE DATA

NODAL DATA

NODE	BOUNDARY COND. THETA/Y/X	COORDINATES X	Y	FORCES FX	FY	FTHETA
1	1	0.00	0.00	.00E+00	.00E+00	.00E+00
2	0	100.00	0.00	.00E+00	.00E+00	.00E+00
3	0	200.00	0.00	.00E+00	.00E+00	.00E+00
4	0	300.00	0.00	.00E+00	.00E+00	.00E+00
5	0	400.00	0.00	.00E+00	.00E+00	.00E+00
6	0	500.00	0.00	.00E+00	.00E+00	.00E+00
7	0	600.00	0.00	.00E+00	.00E+00	.00E+00
8	0	750.00	0.00	.00E+00	.00E+00	.00E+00
9	0	900.00	0.00	.00E+00	.00E+00	.00E+00
10	0	1050.00	0.00	.00E+00	.00E+00	.00E+00
11	0	1200.00	0.00	.00E+00	.00E+00	.00E+00
12	0	1350.00	0.00	.00E+00	.00E+00	.00E+00
13	0	1500.00	0.00	.00E+00	.00E+00	.00E+00
14	0	1700.00	0.00	.00E+00	.00E+00	.00E+00
15	0	1900.00	0.00	.00E+00	.00E+00	.00E+00
16	0	2100.00	0.00	.00E+00	.00E+00	.00E+00
17	0	2300.00	0.00	.00E+00	.00E+00	.00E+00
18	0	2500.00	0.00	.00E+00	.00E+00	.00E+00
19	0	2700.00	0.00	.00E+00	.00E+00	.00E+00
20	0	3000.00	0.00	.00E+00	.00E+00	.00E+00
21	0	3300.00	0.00	.00E+00	.00E+00	.00E+00
22	0	3600.00	0.00	.00E+00	.00E+00	.00E+00
23	0	3900.00	0.00	.00E+00	.00E+00	.00E+
24	0	4200.00	0.00	.00E+00	.00E+00	.00E+00
25	0	4500.00	0.00	.00E+00	.00E+00	.00E+00
26	0	4900.00	0.00	.00E+00	.00E+00	.00E+00
27	0	5300.00	0.00	.00E+00	.00E+00	.00E+00
28	0	5700.00	0.00	.00E+00	.00E+00	.00E+00
29	0	6100.00	0.00	.00E+00	.00E+00	.00E+00
30	0	6500.00	0.00	.00E+00	.00E+00	.00E+00
31	0	6900.00	0.00	.00E+00	.00E+00	.00E+00
32	0	7300.00	0.00	.00E+00	.00E+00	.00E+00
33	0	7700.00	0.00	.00E+00	.00E+00	.00E+00
34	0	8100.00	0.00	.00E+00	.00E+00	.00E+00
35	0	8600.00	0.00	.00E+00	.00E+00	.00E+00
36	0	9100.00	0.00	.00E+00	.00E+00	.00E+00
37	0	9600.00	0.00	.00E+00	.00E+00	.00E+00
38	0	10100.00	0.00	.00E+00	.00E+00	.00E+00
39	0	11600.00	0.00	.00E+00	.00E+00	.00E+00
40	0	13100.00	0.00	.00E+00	.00E+00	.00E+00
41	0	14600.00	0.00	.00E+00	.00E+00	.00E+00
42	0	16100.00	0.00	.00E+00	.00E+00	.00E+00
43	0	17600.00	0.00	.00E+00	.00E+00	.00E+00
44	0	19100.00	0.00	.00E+00	.00E+00	.00E+00
45	0	20600.00	0.00	.00E+00	.00E+00	.00E+00
46	0	22100.00	0.00	.00E+00	.00E+00	.00E+00
47	0	23600.00	0.00	.00E+00	.00E+00	.00E+00
48	0	25100.00	0.00	.00E+00	.00E+00	.00E+
49	0	26600.00	0.00	.00E+00	.00E+00	.00E+00
50	0	28100.00	0.00	.00E+00	.00E+00	.00E+00
51	0	29600.00	0.00	.00E+00	.00E+00	.00E+00
52	0	31100.00	0.00	.00E+00	.00E+00	.00E+00
53	0	32600.00	0.00	.00E+00	.00E+00	.00E+00

CCP SUBSIDENCE SIMULATION USING LITERATURE DATA

CONVERGENCE USING 55 OVERBURDEN ELEMENTS

POUNDS-FEET UNITS

PRELIMINARY DATA

NUMBER OF NODES 56
NUMBER OF OVERBURDEN ELEMENTS 55
NUMBER OF PILLAR ELEMENTS 44
NUMBER OF ROOM ELEMENTS 11

NUMBER OF OVERBURDEN MATERIAL TYPES .. 1
NUMBER OF PILLAR MATERIAL TYPES 7
NUMBER OF ROOM MATERIAL TYPES 3

BODY FORCES WILL BE INCLUDED

ITERATIONS STOP AFTER 150 STEPS OR TOLERANCE = .001

54	0	34100.00	0.00	.00E+00	.00E+00	.00E+00
55	0	35600.00	0.00	.00E+00	.00E+00	.00E+00
56	0	36350.00	0.00	.00E+00	.00E+00	.00E+00

FOR BOUNDARY CONDITIONS 1=FIXED, 0=UNFIXED

CCP SUBSIDENCE SIMULATION USING LITERATURE DATA

MATERIAL TYPES FOR PROBLEM

OVERBURDEN ELEMENTS

TYPE	YOUNG'S MODULUS (E)	POISSON'S RATIO	HEIGHT	WEIGHT DENSITY	'Z'-DEPT
1	.950E+08	.220E+00	3300.00	150.000	1.00

PILLAR MATERIAL TYPES

TYPE	COMPRESSIVE STRENGTH	HEIGHT	'X'-WIDTH	WEIGHT DENSITY
1	1025000.00	10.50	100.00	.136E+03
2	1025000.00	10.50	133.33	.136E+03
3	1025000.00	10.50	200.00	.136E+03
4	1025000.00	10.50	400.00	.136E+03
5	1025000.00	10.50	500.00	.136E+03
6	1025000.00	10.50	900.00	.136E+03
7	1025000.00	10.50	1500.00	.136E+03

TYPE	POISSON'S RATIO	POROSITY	POWER STRAIN	LAMBDA
1	.300E+00	0.000	2.000	.100E+02
2	.300E+00	0.000	2.000	.100E+02
3	.300E+00	0.000	2.000	.100E+02
4	.300E+00	0.000	2.000	.100E+02
5	.300E+00	0.000	2.000	.100E+02
6	.300E+00	0.000	2.000	.100E+02
7	.300E+00	0.000	2.000	.100E+02

BENDING IN ROOM AND PILLAR ELEMENTS WILL BE INCLUDED

ROOM MATERIAL TYPES

TYPE	COMPRESSIVE STRENGTH	HEIGHT	'X'-WIDTH	WEIGHT DENSITY
1	0.00	10.50	75.00	.000E+00
2	0.00	10.50	150.00	.000E+00
3	0.00	10.50	300.00	.000E+00

TYPE	POISSON'S RATIO	POROSITY	POWER STRAIN	LAMBDA
1	.000E+00	0.000	0.000	.000E+00
2	.000E+00	0.000	0.000	.000E+00
3	.000E+00	0.000	0.000	.000E+00

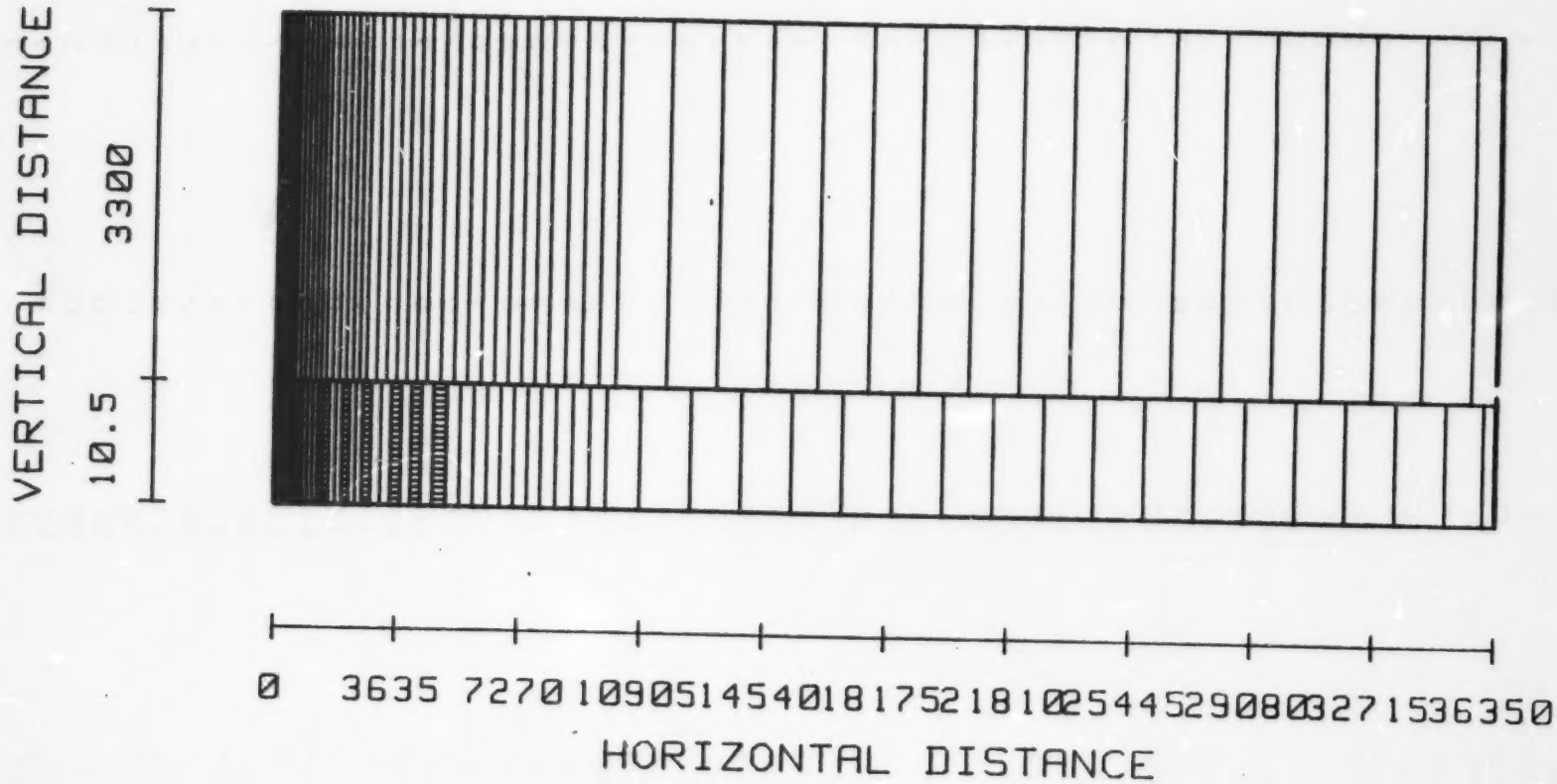
CCP SUBSIDENCE SIMULATION USING LITERATURE DATA
ELEMENT CONNECTIONS

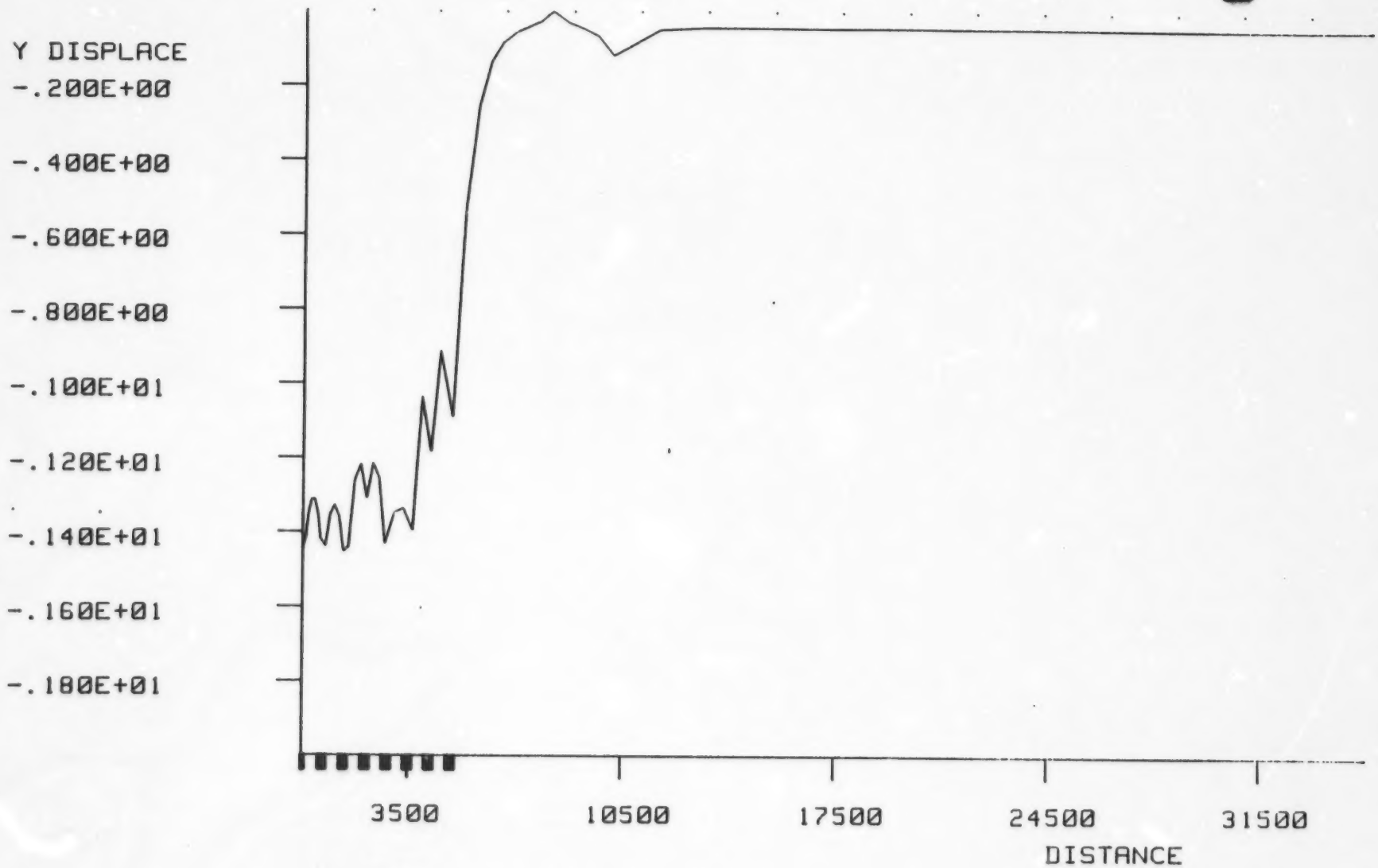
ELEMENT NO.	NODE 1	NODE 2	MATERIAL TYPE	ELEMENT TYPE
1	1	2	1	OB
2	2	3	1	OB
3	3	4	1	OB
4	4	5	1	OB
5	5	6	1	OB
6	6	7	1	OB
7	7	8	1	OB
8	8	9	1	OB
9	9	10	1	OB
10	10	11	1	OB
11	11	12	1	OB
12	12	13	1	OB
13	13	14	1	OB
14	14	15	1	OB
15	15	16	1	OB
16	16	17	1	OB
17	17	18	1	OB
18	18	19	1	OB
19	19	20	1	OB
20	20	21	1	OB
21	21	22	1	OB
22	22	23	1	OB
23	23	24	1	OB
24	24	25	1	OB
25	25	26	1	OB
26	26	27	1	OB
27	27	28	1	OB
28	28	29	1	OB
29	29	30	1	OB
30	30	31	1	OB
31	31	32	1	OB
32	32	33	1	OB
33	33	34	1	OB
34	34	35	1	OB
35	35	36	1	OB
36	36	37	1	OB
37	37	38	1	OB
38	38	39	1	OB
39	39	40	1	OB
40	40	41	1	OB
41	41	42	1	OB
42	42	43	1	OB
43	43	44	1	OB
44	44	45	1	OB
45	45	46	1	OB
46	46	47	1	OB
47	47	48	1	OB
48	48	49	1	OB
49	49	50	1	OB
50	50	51	1	OB
51	51	52	1	OB
52	52	53	1	OB
53	53	54	1	OB
54	54	55	1	OB

55	55	56	1	OB
56	1	0	1	RM
57	2	0	1	RM
58	3	0	1	PR
59	4	0	1	PR
60	5	0	1	PR
61	6	0	1	PR
62	7	0	2	RM
63	8	0	2	RM
64	9	0	2	PR
65	10	0	2	PR
66	11	0	2	PR
67	12	0	2	RM
68	13	0	2	RM
69	14	0	3	PR
70	15	0	3	PR
71	16	0	3	RM
72	17	0	3	PR
73	18	0	3	PR
74	19	0	3	RM
75	20	0	3	PR
76	21	0	3	PR
77	22	0	3	RM
78	23	0	4	PR
79	24	0	3	RM
80	25	0	4	PR
81	26	0	3	RM
82	27	0	4	PR
83	28	0	4	PR
84	29	0	4	PR
85	30	0	4	PR
86	31	0	4	PR
87	32	0	4	PR
88	33	0	4	PR
89	34	0	5	PR
90	35	0	5	PR
91	36	0	5	PR
92	37	0	5	PR
93	38	0	6	PR
94	39	0	7	PR
95	40	0	7	PR
96	41	0	7	PR
97	42	0	7	PR
98	43	0	7	PR
99	44	0	7	PR
100	45	0	7	PR
101	46	0	7	PR
102	47	0	7	PR
103	48	0	7	PR
104	49	0	7	PR
105	50	0	7	PR
106	51	0	7	PR
107	52	0	7	PR
108	53	0	7	PR
109	54	0	7	PR
110	55	0	7	PR

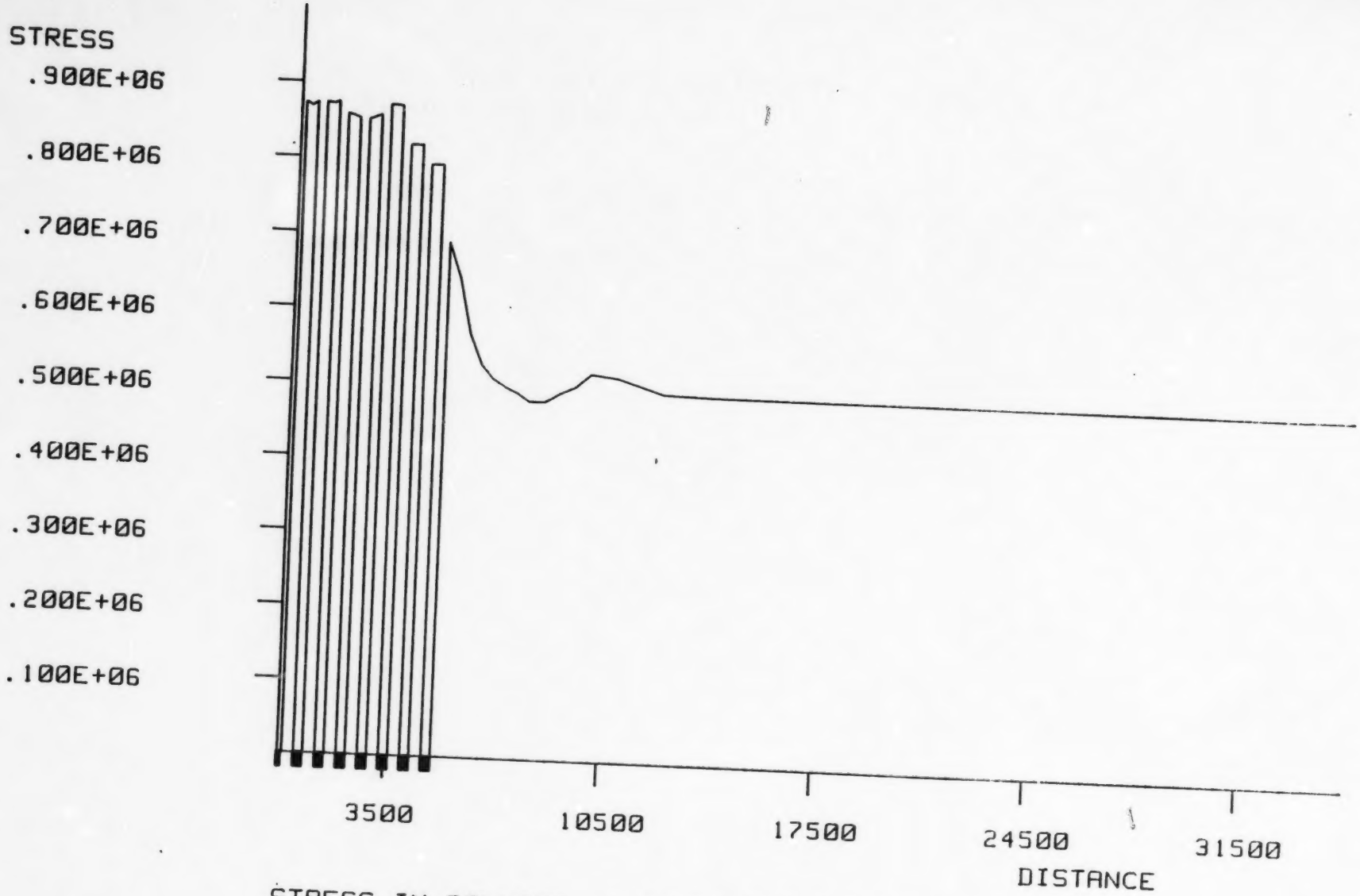
FOR ELEMENT TYPES :
OB = OVERBURDEN; PR = PILLAR; RM = ROOM

CCP SUBSIDENCE SIMULATION USING LITERATURE DATA





SURFACE DISPLACEMENTS
CCP SUBSIDENCE SIMULATION USING LITERATURE DATA



STRESS IN PILLARS -- COMPRESSION POSITIVE
CCP SUBSIDENCE SIMULATION USING LITERATURE DATA

CCP SUBSIDENCE SIMULATION USING LITERATURE DATA

SURFACE DISPLACEMENTS

NODE	VERTICAL-DISPL.	HORIZONTAL DISPL.	ANGULAR DISPL.
1	1.4415E+00	-1.2515E-02	-7.5849E-06
2	1.4177E+00	-1.1819E-02	-7.1499E-06
3	1.3477E+00	-8.2522E-03	-4.9750E-06
4	1.3132E+00	-4.0990E-03	-2.4521E-06
5	1.3137E+00	-2.4269E-04	-1.1290E-07
6	1.3491E+00	2.4357E-03	1.5123E-06
7	1.4199E+00	2.6225E-03	1.6297E-06
8	1.4391E+00	-1.2789E-03	-7.2833E-07
9	1.3541E+00	-2.1238E-03	-1.2340E-06
10	1.3282E+00	-1.1338E-04	-1.2714E-08
11	1.3609E+00	1.5578E-03	1.0031E-06
12	1.4528E+00	-7.0639E-04	-3.6297E-07
13	1.4402E+00	-6.6748E-03	-3.9741E-06
14	1.2613E+00	-2.8112E-03	-1.6243E-06
15	1.2189E+00	-9.8635E-05	1.7826E-08
16	1.3102E+00	-2.6163E-04	-8.2160E-08
17	1.2164E+00	-2.7193E-04	-8.9602E-08
18	1.2561E+00	2.2269E-03	1.4235E-06
19	1.4320E+00	4.4531E-03	2.7812E-06
20	1.3482E+00	-7.1219E-04	-3.3658E-07
21	1.3368E+00	-1.2535E-03	-6.5416E-07
22	1.3960E+00	-1.4710E-02	-8.8043E-06
23	1.0379E+00	-6.6948E-04	-2.8908E-07
24	1.1846E+00	-6.3092E-03	-3.7057E-06
25	9.1395E-01	-5.4266E-04	-2.0944E-07
26	1.0919E+00	-2.4301E-02	-1.4610E-05
27	5.2633E-01	-8.9777E-04	-4.2748E-07
28	2.5745E-01	-3.7459E-04	-1.2807E-07
29	1.3584E-01	-2.1525E-04	-5.1159E-08
30	8.1755E-02	-1.3802E-04	-2.1900E-08
31	5.6694E-02	-9.5849E-05	-1.0721E-08
32	4.2122E-02	-7.2397E-05	-7.9055E-09
33	2.7060E-02	-6.2480E-05	-1.0978E-08
34	0.0000E+00	-3.0845E-05	4.4351E-10
35	3.0719E-02	-9.5660E-06	6.2818E-09
36	4.7043E-02	-3.7724E-06	5.1331E-09
37	6.8434E-02	9.4339E-06	1.0092E-08
38	1.1994E-01	-5.8467E-06	-3.8935E-10
39	4.8594E-02	-2.7175E-06	-3.6460E-10
40	4.3208E-02	-8.1910E-07	-2.7407E-11

41	4.2802E-02	-2.8081E-07	-2.0715E-12
42	4.2772E-02	-9.9266E-08	-1.5894E-13
43	4.27691 -02	-3.5280E-08	-1.3022E-14
44	4.2769E-02	-1.2430E-08	-1.3514E-15
45	4.2769E-02	-4.0257E-09	-2.1900E-16
46	4.2769E-02	-2.9582E-10	-1.6213E-17
47	4.2769E-02	3.0905E-09	1.1690E-16
48	4.2769E-02	1.0065E-08	3.8190E-16
49	4.2769E-02	2.8725E-08	1.0899E-15
50	4.2769E-02	8.0736E-08	3.0623E-15
51	4.2769E-02	2.2648E-07	8.5762E-15
52	4.2769E-02	6.3518E-07	2.3861E-14
53	4.2769E-02	1.7813E-06	6.4374E-14
54	4.2769E-02	4.9960E-06	3.8241E-13
55	4.2770E-02	3.5712E-05	1.3153E-08
56	1.4348E+00	3.0282E-01	1.8352E-04

APPENDIX 10

SUBSIDENCE QUESTIONNAIRE

REGIONAL MINE STABILITY PROJECT

SUBSIDENCE QUESTIONNAIRE

SUBSIDENCE QUESTIONNAIRE

REGIONAL MINE STABILITY PROJECT

S1. At what elevation below surface is the mining horizon?

Highest _____

Lowest _____

Average _____

S2. On what date did your mine commence underground operations?

S3. Please list the dates of EACH survey taken, including the initial survey, at your site, to include the time period during which the survey was performed. Please also indicate the frequency of measurements taken, and any comment on the validity or acceptability of each survey.

S4. To what order of precision was each survey conducted (First order, Second order, etc.)?

S5. For EACH of the surveys conducted, would you please provide the following:

a) The location of the datum in relation to the survey.

b) Map showing location of survey lines and stations.

c) Mine plans showing the time rate of mining advance relative to the subsidence surveys.

d) All available subsidence survey records of vertical deformation (i.e. movement or displacement), horizontal deformation, or both, for each subsidence station and/or monument.

e) Constructed subsidence profiles relating to mining rate, sequencing or time.

f) Equipment used.

g) Calculated strain and/or tilt data.

h) Mining panel and pillar dimensions and extraction ratio for areas relevant to each subsidence profile.

Note: Where dimensions of panels and/or pillars changed between surveys, please indicate the time and dimensions of the changes.

Please see question 19 re: extraction ratio.

S6. What is the maximum amount of subsidence that has occurred?

S7. In your opinion, does the subsidence record correlate with the mining sequence and/or extraction ratio?

Yes _____ No _____

Note: See also question S19.

S8. If answer to S7 is affirmative, is corroborating data available?

Yes _____ No _____

If affirmative, please supply same.

S9. What relationship is there between underground closure of the mine openings and surface subsidence?

S10. Can you provide corroborating data between surface subsidence and underground closure?

Yes _____ No _____

If affirmative, please supply.

S11. Please describe any visual effects of surface subsidence. (e.g. structural damage, drainage interference, etc...).

S12. Please describe any effects of surface subsidence on your surface and/or underground operations (e.g. tailings pile, pond, dikes, pipelines, etc...).

S13. a) In your mined out (abandoned) areas, do you have closure stations still accessible for measurement?

Yes _____ No _____

b) If answer is affirmative, please provide data on closure (magnitude and rate) relative to each survey carried out.

S14. Please provide the following information/data:

a) relevant geological (stratigraphic) sections and structural data for each subsidence profile.

If this information is NOT available, please provide the following:

i) geologic logs for shaft pilot holes, and/or

ii) geologic logs and sections from shaft sinking.

iii) contour maps and seismic sections from seismic surveys covering the mining area, to illustrate changes in thickness of overlying formations.

S15. Please provide any material on measurements of physical parameters for the overburden and the evaporite formations.

S16. Please provide the design parameters used for determining the shaft pillar size.

S17. To what depths are the survey monuments installed.

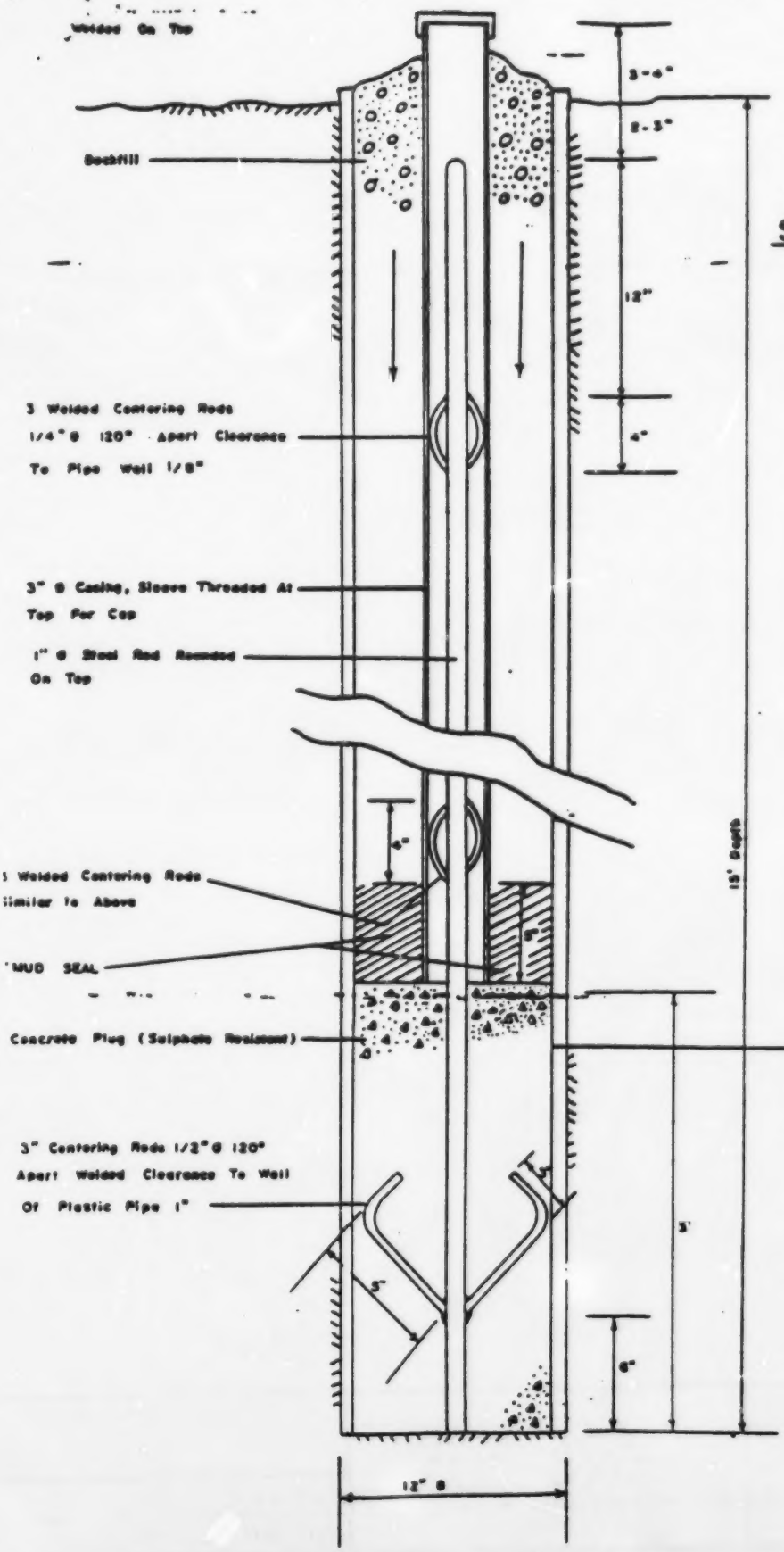
S18. Please provide diagrams of monuments used in each survey.

S19. Please define the term "Extraction Ratio" as used when answering questions S5.h) and S7.

APPENDIX 11

CURRENT DESIGN OF SUBSIDENCE MONUMENTS

QUESTION S 18.



MODIFIED

SUBSIDENCE MONUMENT

FIG IV-1

MATERIALS

1 -	3" Schedule 40 Pipe 13'-3" Long
1 -	1" Ø Steel Rod 14' Long
1 -	Pipe Cap With Welded 1" Nut
3 -	10" Long Centering Rod 1/2" Ø
6 -	4 1/2" Long Centering Rod 1 1/4" Ø
8	C.F. Concrete Sulphate Resistant

Drawn By - J Bell

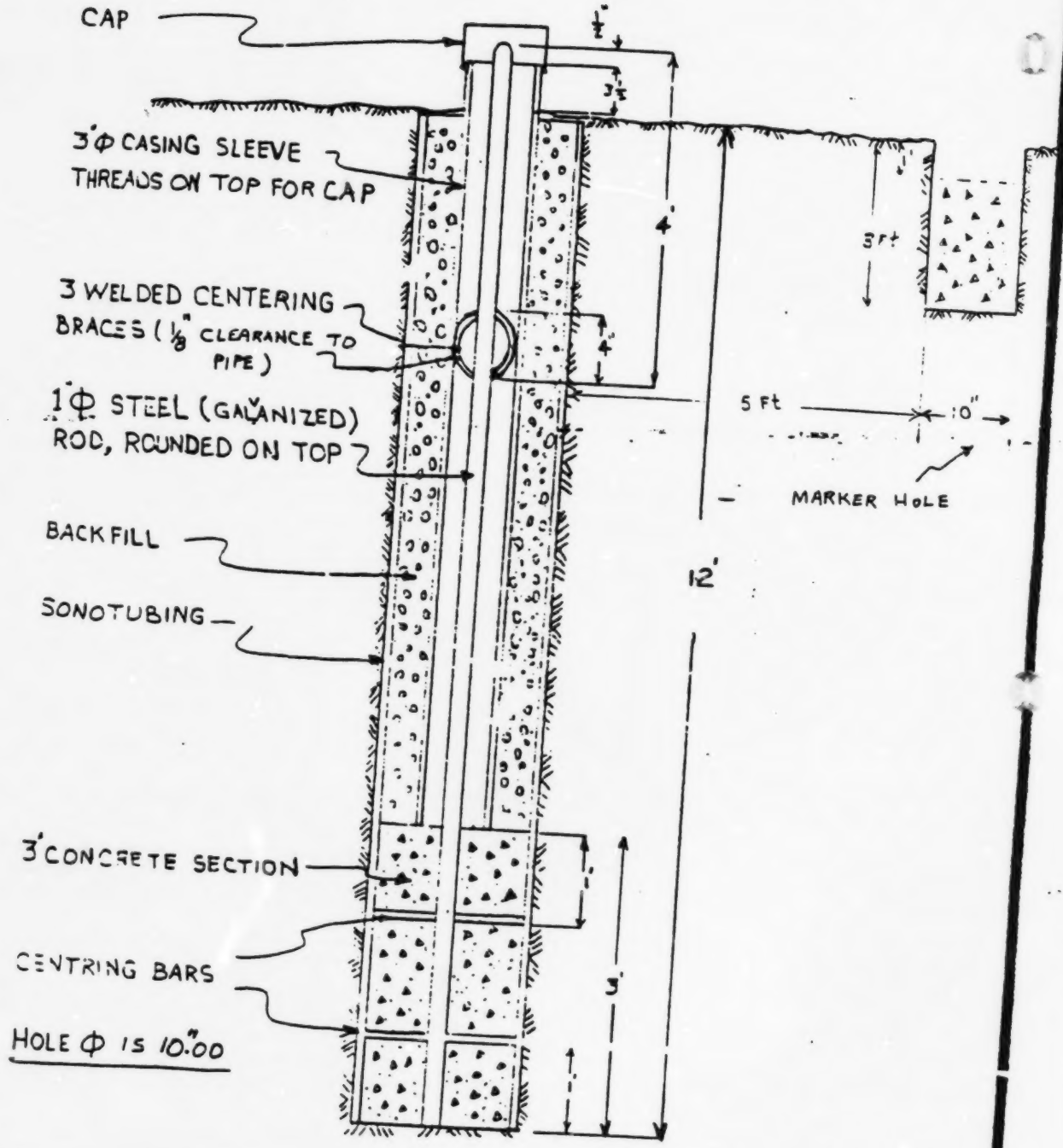
DATE JAN 1977

FIG IV-1

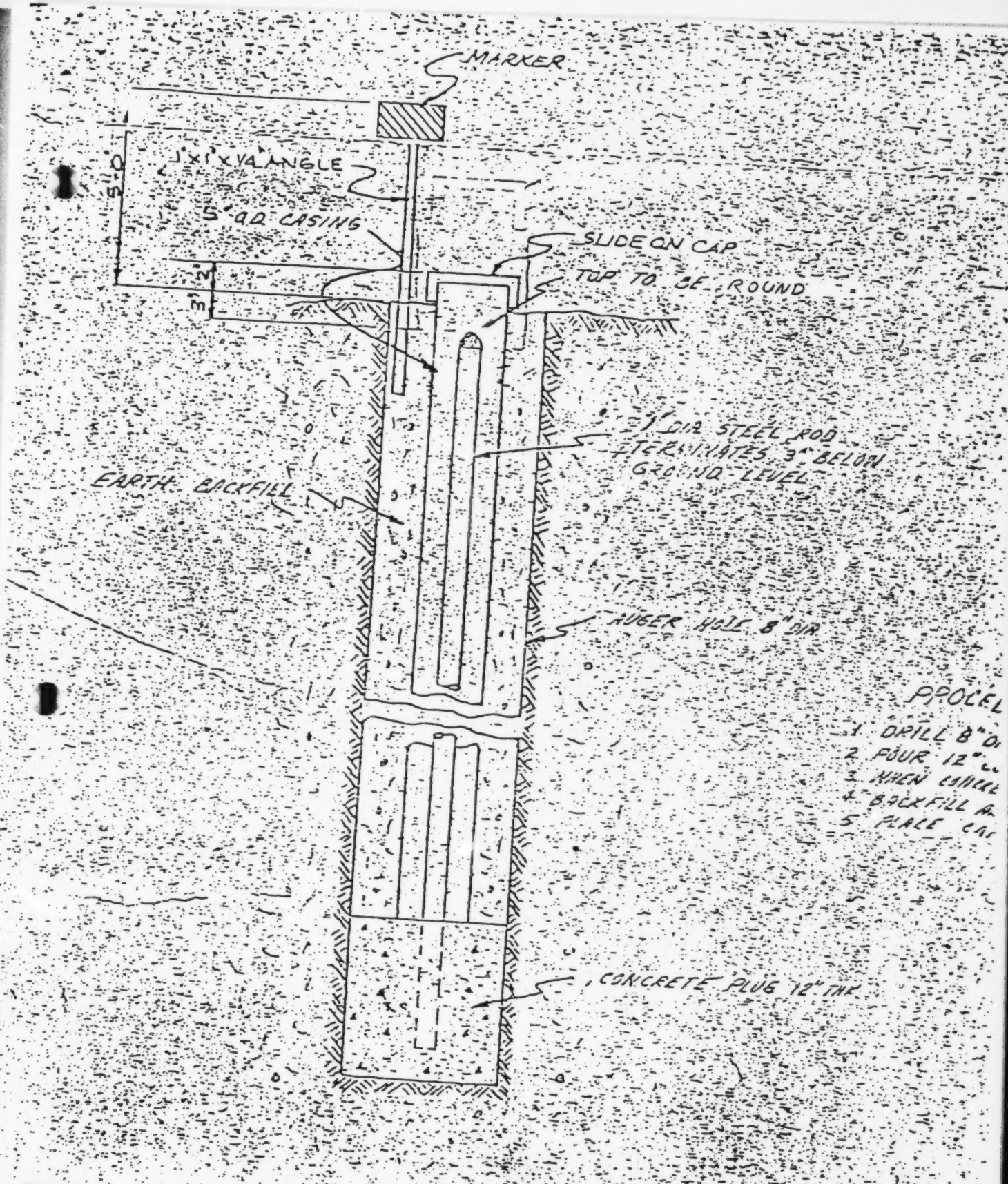
Potash Corporation of Saskatchewan

ROCANVILLE DIVISION

QUESTION S 18.

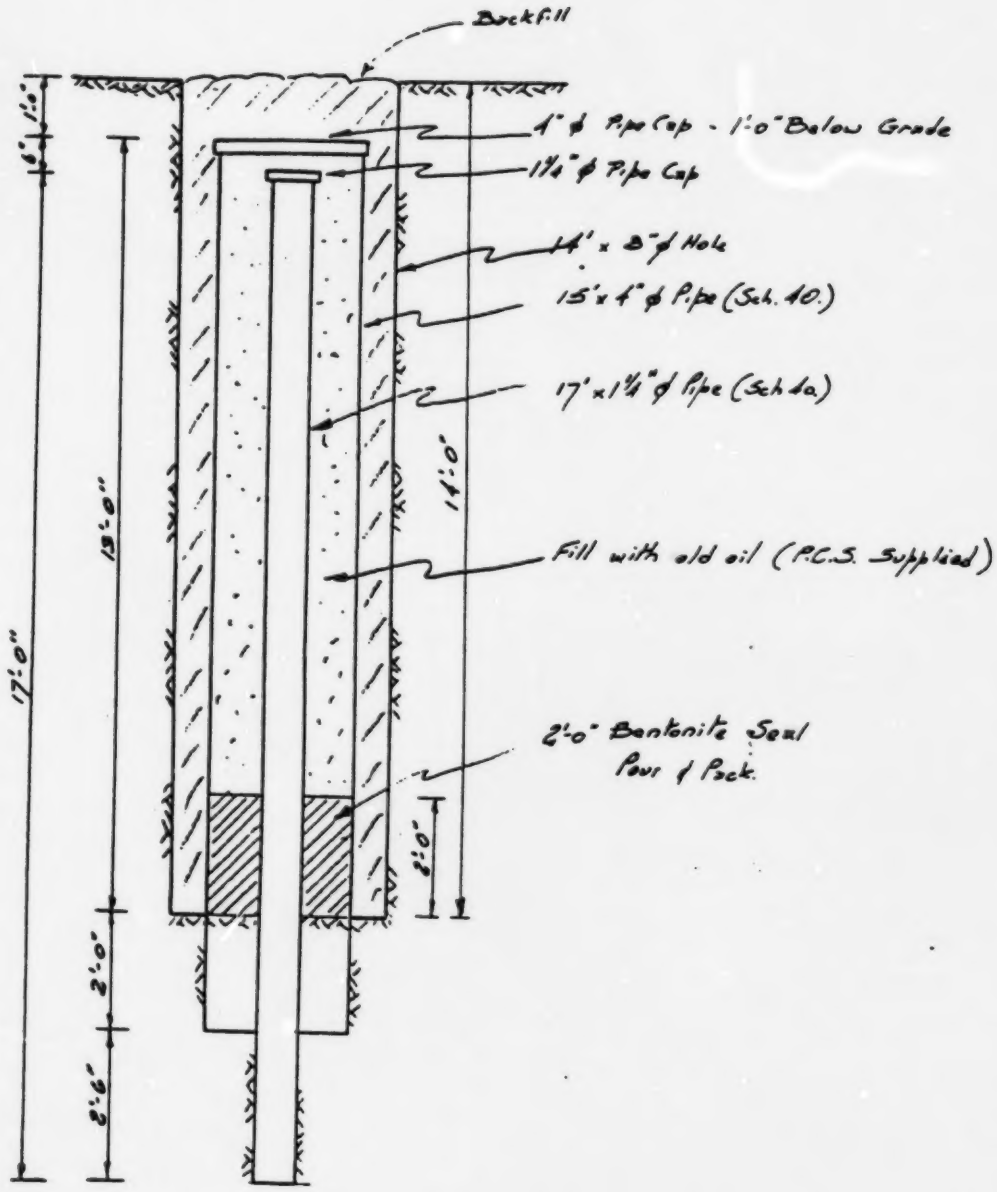


REV. 1. HOLE Φ DECREASED, STEEL ROD TO STICK OUT OF CASING. NUMBER OF CENTRING BRACES REDUCED, HOLE 2. LENGTH REDUCED & MODIFIED CENT. BARS	MINING LIMITED — ROCANVILLE DIVISION	SURFACE SUBSIDENCE BENCH MARKS
DRWN: M. A. MOLAVI, FEB 1983		DWG No.: 1
CHK'D:		SHT No. ___ of ___



SUBSIDENCE B.M.
DETAIL

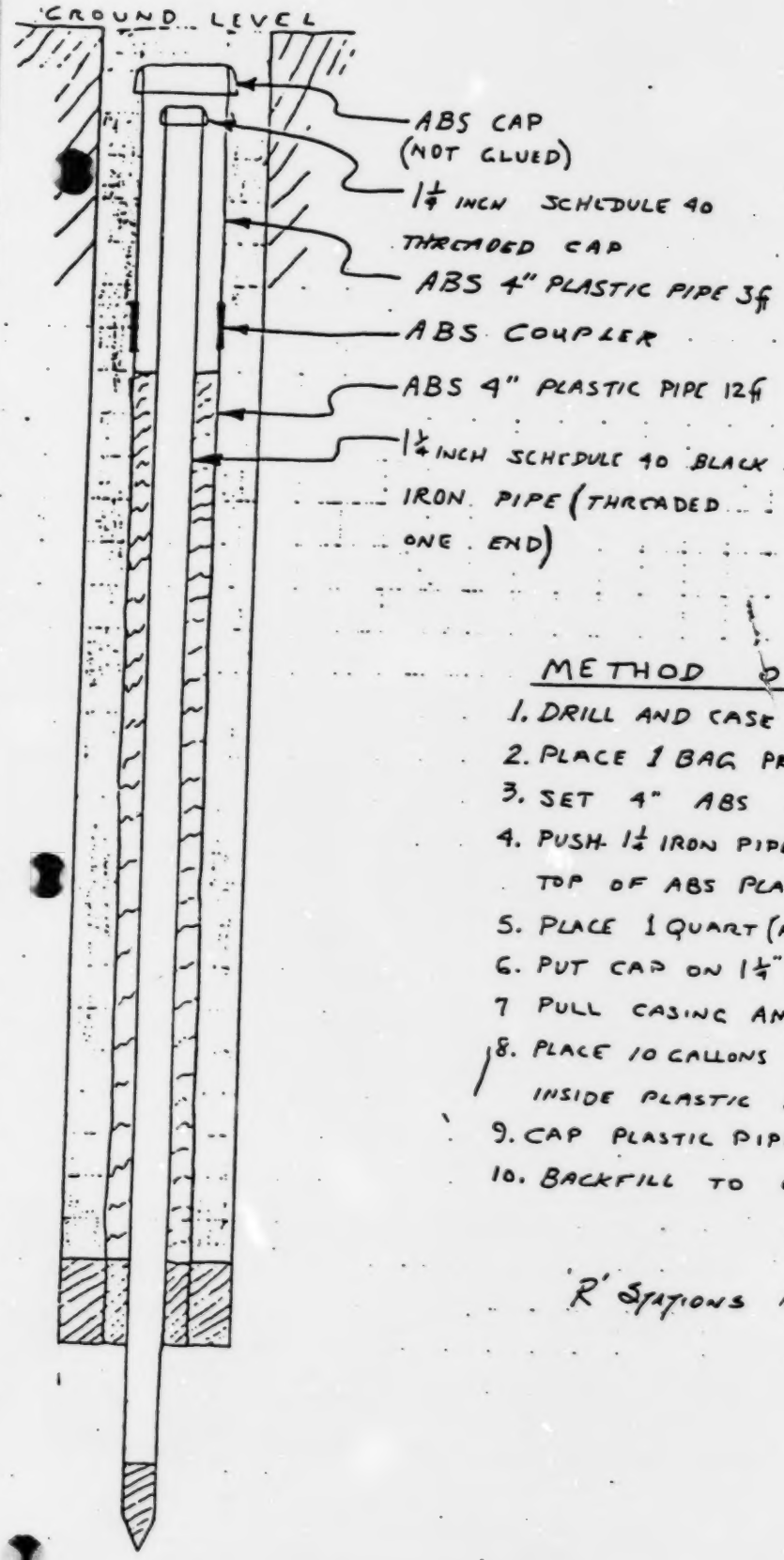
FIG. 5.



S 18a.

The Potash Corporation of Saskatchewan MINING LIMITED	LANIGAN DIVISION
SUBSIDENCE STATIONS INSTALLED 1978 & 1979	
DRAWN BY: G.A.M.	SCALE : N.T.S.
CHECK BY: H.D.G.	DATE : 3-9-80
APPVD. BY H.D.G.	DWG. NO. M.A. 36

SUBSIDENCE BENCH MARK DETAIL



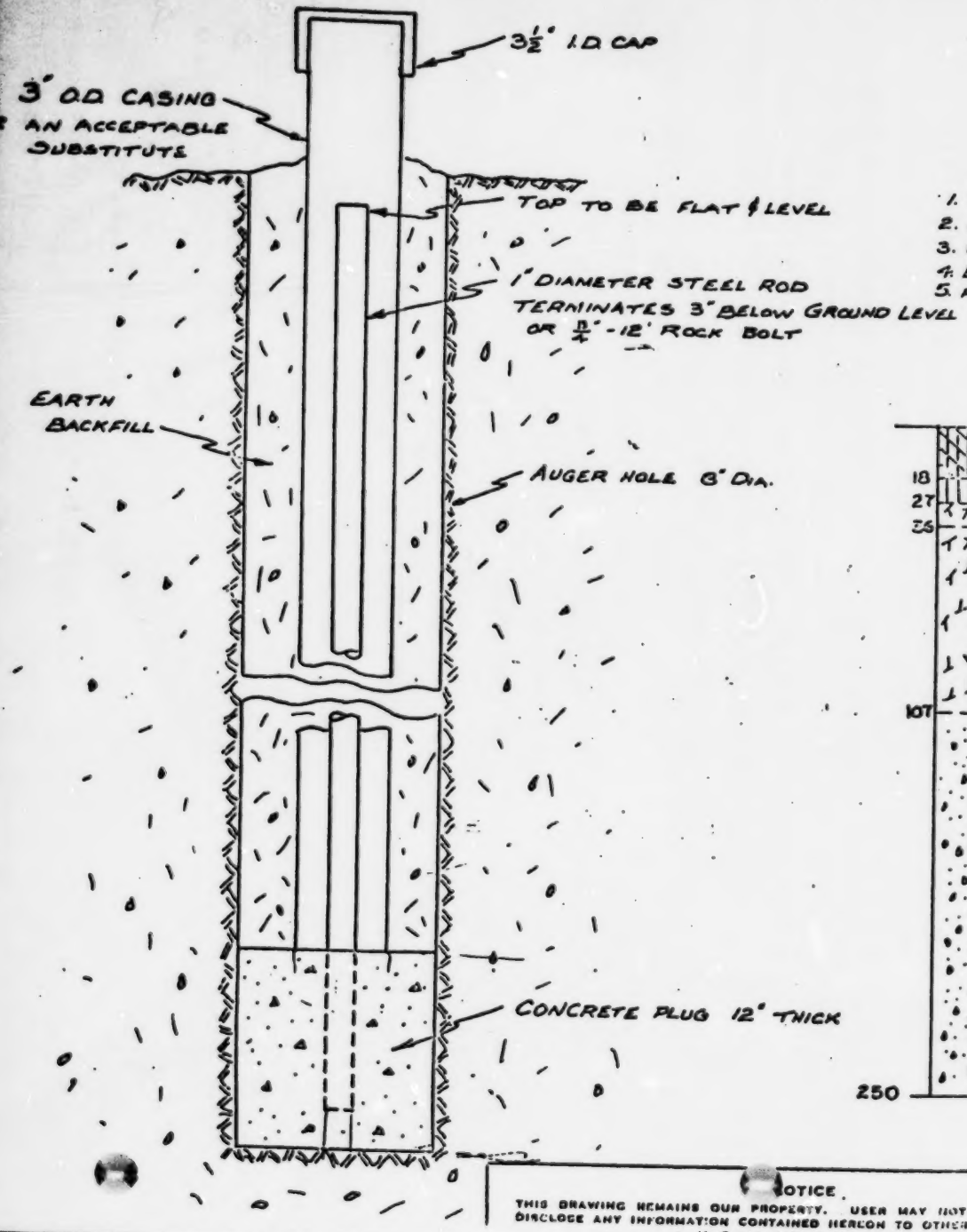
METHOD OF INSTALLATION

1. DRILL AND CASE 10"-12" HOLE 15½ ft DEEP.
2. PLACE 1 BAG PREMIX MORTAR (ZERO SLUMP)
3. SET 4" ABS PLASTIC PIPE TO 15½ ft DEEP
4. PUSH 1¼ IRON PIPE UNTIL TOP IS ½ ft BELOW TOP OF ABS PLASTIC. PIPE (USE 1⅛ STIFFENER ROD)
5. PLACE 1 QUART (APPROX) GROUT INSIDE 1¼" IRON PIPE
6. PUT CAP ON 1¼" IRON PIPE.
7. PULL CASING AND BACK FILL OUTSIDE PLASTIC PIPE.
8. PLACE 10 GALLONS SLURRY MIXED WITH 1 BAG BENTONITE INSIDE PLASTIC PIPE
9. CAP PLASTIC PIPE
10. BACKFILL TO GROUND LEVEL.

'R' STATIONS INSTALLED 1980 (SEE S8b)

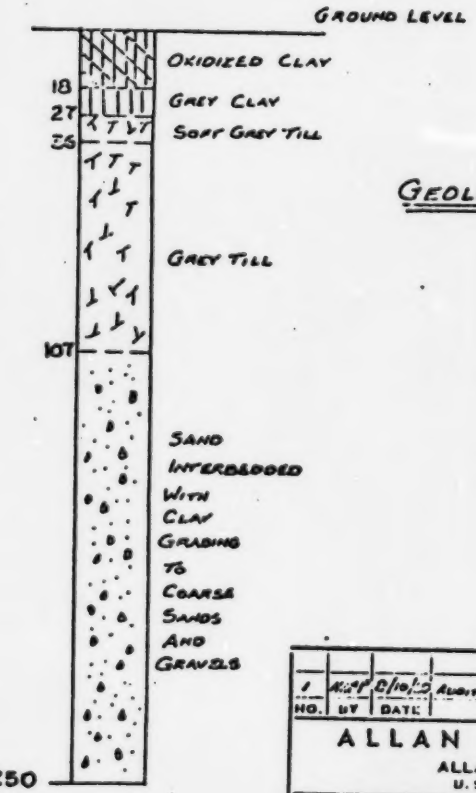
P.C.S. LANIGAN
SUBSIDENCE B.M.'S
DEC 1 - 1980
SCALE: 1"=2' DWG: 1
LEGEND
MORTAR (ZERO SLUMP)
1 BAG BENTONITE (10 GAL.)

SCHEDULE "B"



PROCEDURE

1. DRILL 8" DIA. AUGER HOLE TO DESIRED DEPTH (12')
2. POUR 12" CONCRETE PLUG & SET 1" STEEL ROD.
3. WHEN CONCRETE HAS SET - PLACE 3" PIPE OVER ROD.
4. BACKFILL AROUND PIPE & LIGHTLY TAMP
5. PLACE CAP ON CASING.



GEOLOGICAL LOG

FIG. 8

NO.	BY	DATE	REVISION	APP'D
1	K.A.P.	2/10/60	ADDITION OF GEOLOGICAL LOG.	

ALLAN POTASH MINES
ALLAN, SASKATCHEWAN
U.S. BORAX, OPERATOR

SURFACE SUBSIDENCE B.M.'S
DESIGN DETAILS

C.P. NO.	PROJ. NO.	PRINTED	
SCALE N.S.	CHECKED	DATE	
DRAWN K.A.P.	DATE 7/10/59	APP'D.	DATE
DWG. NO.			REVISION

NOTICE
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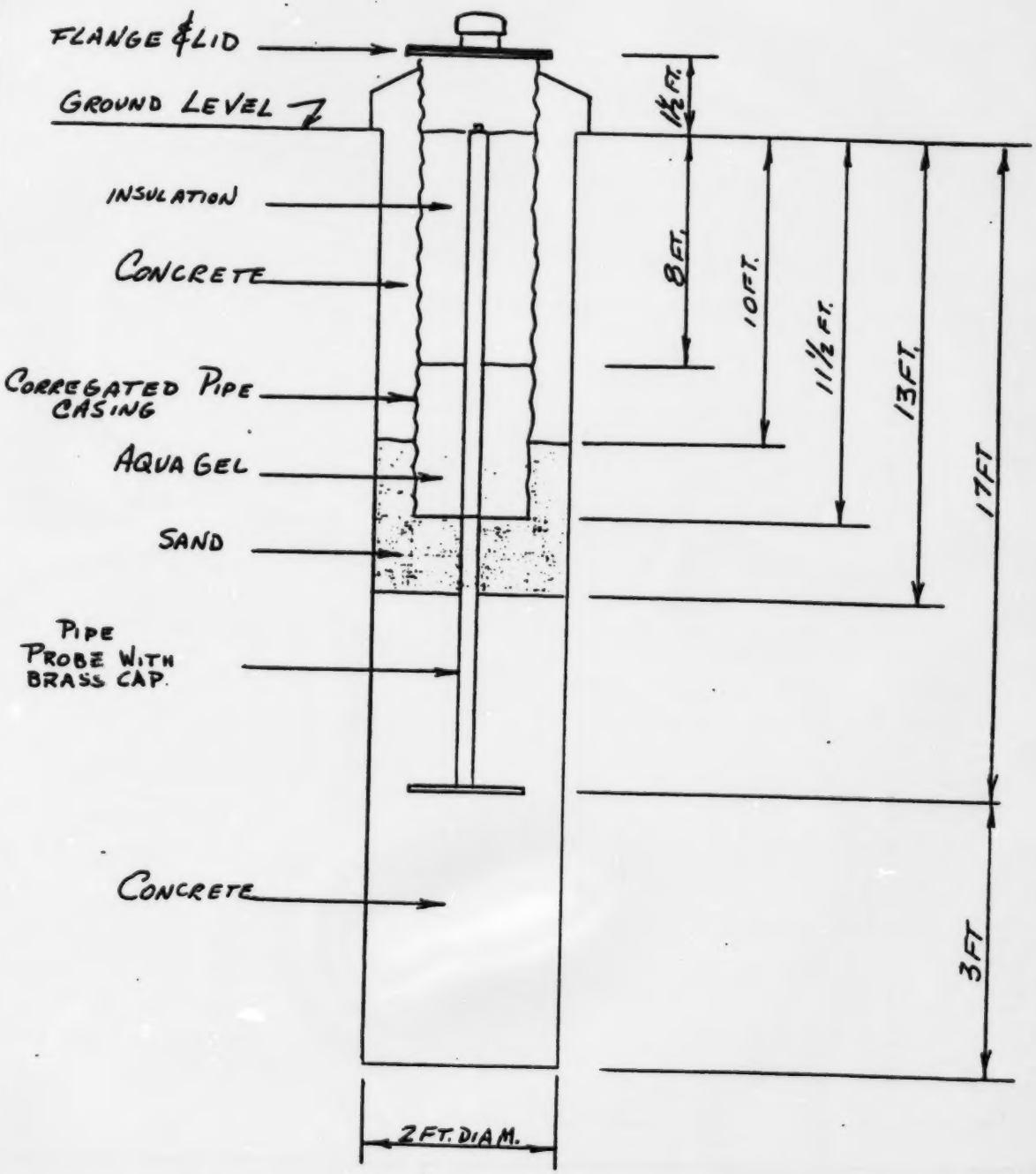
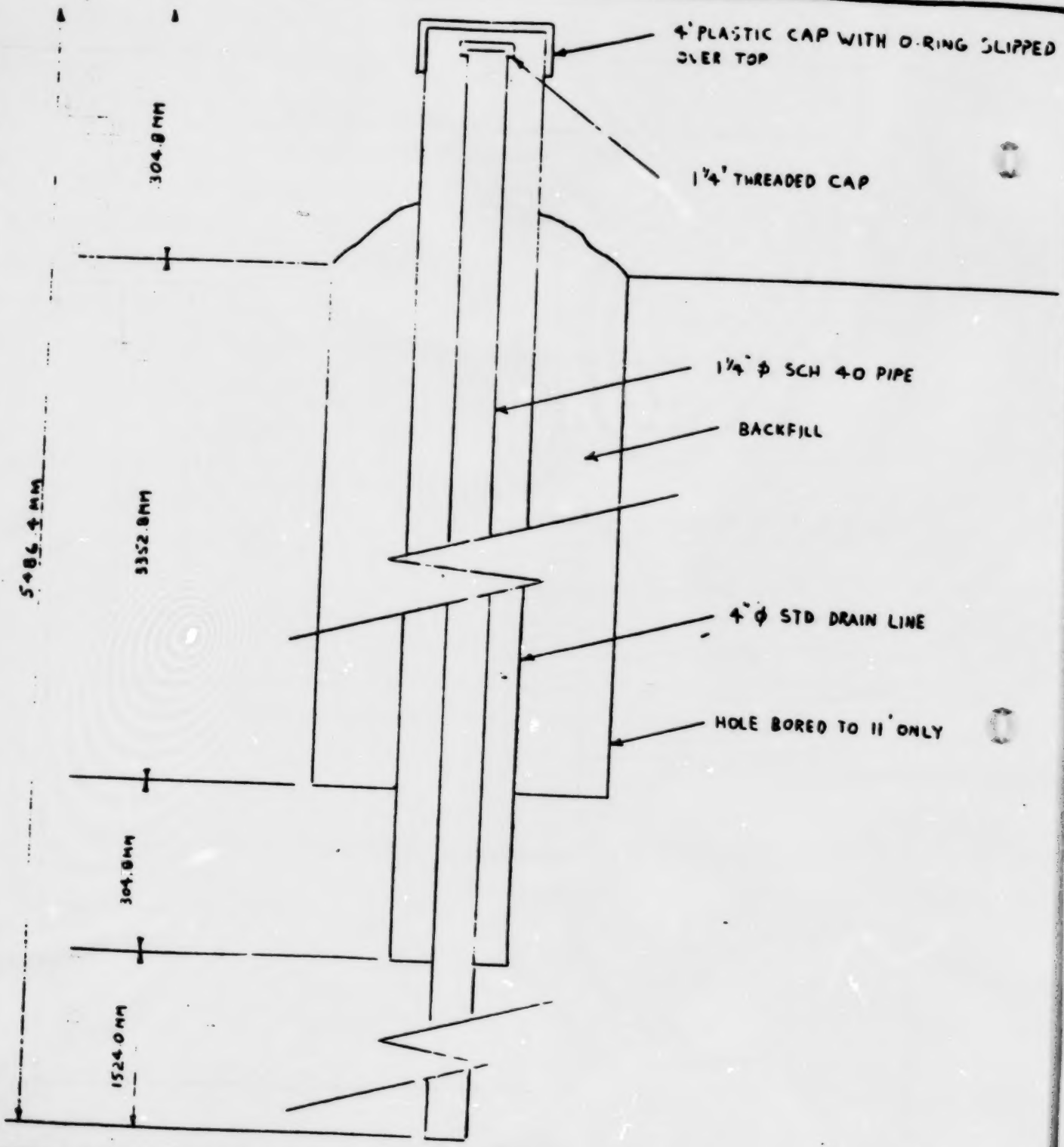
SUBSIDENCE STATION DETAIL



DUVAL CORPORATION OF CANADA SASKATOON, SASKATCHEWAN		PROJECT	
		EQUIP. NO.	
		SUBJECT	
BY T.Z.W.	DATE OCT 19/72	PAGE OF	TAG S-18



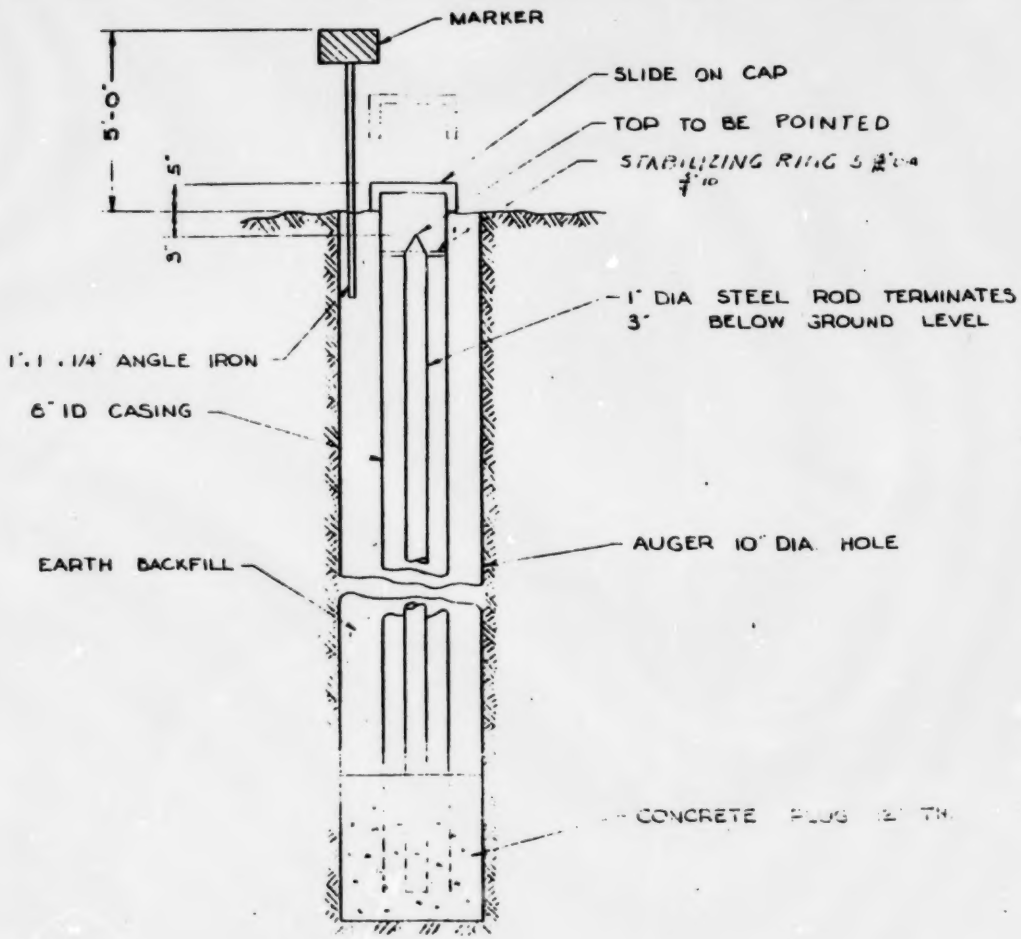
NOTES

P.C.S. Mining — Cory Division

SURFACE SUBSIDENCE STATION (1980)

Dwn. R.A.D			
Chk.	54/80/52	Scale N.T.S.	SK 19-590

11-5D-0162A



DETAIL (SUBSIDENCE B.M.)

PROCEDURE

1. DRILL 10" DIA. AUGER HOLE TO DESIRED DEPTH (12'-0")
2. POUR 12" CONCRETE PLUG & SET 1 DIA STEEL ROD (CENTERED)
3. WHEN CONCRETE HAS SET, PLACE 6" DIA PIPE OVER ROD
4. BACKFILL AROUND PIPE
5. PLACE CAP ON CASING
6. INSTALL MARKER

NO	REVISION	DATE	BY

CENTRAL CANADA POTASH

DRAWN BY W. SCHICK
CHECKED BY
APPROVED BY
SCALE
DATE OCT 21/81

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